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TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS.

VOL. XVIII.

MAY, 1889, TO FEBRUARY, 1890,

INCLUSIVE.

NEW YORK CITY:
PUBLISHED BY THE INSTITUTE,
AT THE OFFICE OF THE SECRETARY.
1890.

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PREFACE.

In consequence of the large space required in this volume for Proceedings and Papers, the list of members has been omitted. Such lists are published frequently in pamphlet form and distributed to members, and extra copies may be had at any time from the Secretary, on the terms stated in this volume, under the head of Publications.

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---

* Abram S. Hewitt, New York City, was elected President in February, 1890.

† Messrs. Henry M. Howe, Boston, Mass.; S. F. Emmons, Washington, D. C., and H. S. Munroe, New York City, were elected Vice-Presidents in February, 1890.

‡ Messrs. Joseph D. Weeks, Pittsburgh, Pa.; W. J. Taylor, Philadelphia, Pa., and S. T. Wellman, Cleveland, Ohio, were elected Managers in February, 1890.

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LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO JUNE, 1890.

Number.	Place.	Date.	Transactions.
I.	Wilkes-Barre, Pa.,*	May, 1871,	i. 3
II.	Bethlehem, Pa.,	August, 1871,	i. 10
III.	Troy, N. Y.,	November, 1871,	i. 13
IV.	Philadelphia, Pa.,	February, 1872,	i. 17
V.	New York, N. Y.,*	May, 1872,	i. 20
VI.	Pittsburgh, Pa.,	October, 1872,	i. 25
VII.	Boston, Mass.,	February, 1873,	i. 28
VIII.	Philadelphia, Pa.,*	May, 1873,	ii. 3
IX.	Easton, Pa.,	October, 1873,	ii. 7
X.	New York, N. Y.,	February, 1874,	ii. 11
XI.	St. Louis, Mo.,*	May, 1874,	iii. 3
XII.	Hazleton, Pa.,	October, 1874,	iii. 8
XIII.	New Haven, Conn.,	February, 1875,	iii. 15
XIV.	Dover, N. J.,*	May, 1875,	iv. 3
XV.	Cleveland, O.,	October, 1875,	iv. 9
XVI.	Washington, D. C.,	February, 1876,	iv. 18
XVII.	Philadelphia, Pa.,†	June, 1876,	v. 3
XVIII.	Philadelphia, Pa.,	October, 1876,	v. 19
XIX.	New York, N. Y.,	February, 1877,	v. 27
XX.	Wilkes-Barre, Pa.,*	May, 1877,	vi. 3
XXI.	Amenia, N. Y.,	October, 1877,	vi. 10
XXII.	Philadelphia, Pa.,	February, 1878,	vi. 18
XXIII.	Chattanooga, Tenn.,*	May, 1878,	vii. 3
XXIV.	Lake George, N. Y.,	October, 1878,	vii. 103
XXV.	Baltimore, Md.,*	February, 1879,	vii. 217
XXVI.	Pittsburgh, Pa.,	May, 1879,	viii. 3
XXVII.	Montreal, Canada,	September, 1879,	viii. 121
XXVIII.	New York, N. Y.,*	February, 1880,	viii. 275
XXIX.	Lake Superior, Mich.,	August, 1880,	ix. 1
XXX.	Philadelphia, Pa.,*	February, 1881,	ix. 275
XXXI.	Staunton, Va.,	May, 1881,	x. 1
XXXII.	Harrisburg, Pa.,	October, 1881,	x. 119
XXXIII.	Washington, D. C.,*	February, 1882,	x. 225
XXXIV.	Denver, Col.,	August, 1882,	xi. 1
XXXV.	Boston, Mass.,*	February, 1883,	xi. 217
XXXVI.	Roanoke, Va.,	June, 1883,	xii. 3
XXXVII.	Troy, N. Y.,	October, 1883,	xii. 175
XXXVIII.	Cincinnati, O.,*	February, 1884,	xii. 447
XXXIX.	Chicago, Ill.,	May, 1884,	xiii. 1
XL.	Philadelphia, Pa.,	September, 1884,	xiii. 285
XLI.	New York, N. Y.,*	February, 1885,	xiii. 585
XLII.	Chattanooga, Tenn.,	May, 1885,	xiv. 1
XLIII.	Halifax, N. S.,	September, 1885,	xiv. 307
XLIV.	Pittsburgh, Pa.,*	February, 1886,	xiv. 587
XLV.	Bethlehem, Pa.,	May, 1886,	xv. lxiii.
XLVI.	St. Louis, Mo.,	October, 1886,	xv. lxx.
XLVII.	Scranton, Pa.*	February, 1887,	xv. lxxvii.
XLVIII.	Utah and Montana,	July, 1887,	xvi. xvii.
XLIX.	Duluth, Minn.,	July, 1887,	xvi. xxiv.
L.	Boston, Mass.,*	February, 1888,	xvi. xxviii.
LI.	Birmingham, Ala.,	May, 1888,	xvii. xix.
LII.	Buffalo, N. Y.,	October, 1888,	xvii. xxiv.
LIII.	New York, N. Y.,*	February, 1889,	xvii. xxxi.
LIV.	Colorado,	June, 1889,	xviii. xvii.
LV.	Ottawa, Canada,	October, 1889,	xviii. xxiv.
LVI.	Washington, D. C.,*	February, 1890,	xviii. xxx.

* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting, May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

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THE publications of the Institute comprise:

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3. Annual volumes of *Transactions*, containing the list of officers, rules, etc.; the Proceedings; and the papers, *revised for final publication.* (In this revision after the preliminary publication, authors are permitted to use the largest liberty; and the changes and additions made in papers are sometimes important. It should be borne in mind, by those who study or quote a paper in the preliminary edition, that they may not have in that form the ultimate and deliberate expression of the author's views. It should be added, however, that in the majority of cases there is no essential change, the correction of typographical errors and additions of later information being the usual alterations.)

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RULES

ADOPTED MAY, 1873. AMENDED MAY, 1875, 1877, AND 1878, FEBRUARY, 1880, 1881, 1887, AND 1890.

I.

OBJECTS.

THE objects of the AMERICAN INSTITUTE OF MINING ENGINEERS are to promote the arts and sciences connected with the economical production of the useful minerals and metals, and the welfare of those employed in these industries, by means of meetings for social intercourse, and the reading and discussion of professional papers, and to circulate, by means of publications among its members and associates, the information thus obtained.

II.

MEMBERSHIP.

The Institute shall consist of Members, Honorary Members, and Associates. Members and Honorary Members shall be professional mining engineers, geologists, metallurgists, or chemists, or persons practically engaged in mining, metallurgy, or metallurgical engineering. Associates shall include all suitable persons desirous of being connected with the Institute, and duly elected as hereinafter provided. Each person desirous of becoming a member or associate shall be proposed by at least three members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) upon receiving three-fourths of the votes cast, and shall become a member or associate on the payment of his first dues. Each person proposed as an honorary member shall be recommended by at least ten members or associates, approved by the Council, and elected by ballot at a regular meeting (or by ballot at any time conducted through the mail, as the Council may prescribe) on receiving nine-tenths of the votes cast; *Provided*, that the number of honorary members shall not exceed twenty. The Council may at any time change the classification of a person elected as associate, so as to make him a member, or *vice versa*, subject to the approval of the Institute. All members and associates shall be equally entitled to the privileges of membership; *Provided*, that honorary members shall not be entitled to vote.

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Any member or associate may be stricken from the list on recommendation of the Council, by the vote of three-fourths of the members and associates present at any annual meeting, due notice having been mailed in writing by the Secretary to the said member or associate.

III.

DUES.

The dues of members and associates shall be ten dollars, payable upon their election, and ten dollars per annum thereafter, payable in advance at the annual meeting. Honorary members shall not be liable to dues. Any member or associate not in arrears may become by the payment of one hundred dollars at one time a life-member or associate, and shall not be liable thereafter to annual dues. Any member or associate in arrears may, at the discretion of the Council, be deprived of the receipt of publications, or stricken from the list of members when in arrears for one year; *Provided*, that he may be restored to membership by the Council on payment of all arrears, or by re-election after an interval of three years.

IV.

OFFICERS.

The affairs of the Institute shall be managed by a Council, consisting of a President, six Vice-Presidents, nine Managers, a Secretary and a Treasurer, who shall be elected from among the members and associates of the Institute at the annual meetings, to hold office as follows:

The President, the Secretary, and the Treasurer for one year (and no person shall be eligible for immediate re-election as President who shall have held that office subsequent to the adoption of these rules, for two consecutive years), the Vice-Presidents for two years, and the Managers for three years; and no Vice-President or Manager shall be eligible for immediate re-election to the same office at the expiration of the term for which he was elected. At each annual meeting a President, three Vice-Presidents, three Managers, a Secretary and a Treasurer shall be elected, and the term of office shall continue until the adjournment of the meeting at which their successors are elected.

The duties of all officers shall be such as usually pertain to their offices, or may be delegated to them by the Council or the Institute; and the Council may in its discretion require bonds to be given by the Treasurer. At each annual meeting the Council shall make a report of proceedings to the Institute, together with a financial statement.

Vacancies in the Council may occur by death or resignation; or the Council may, by a vote of the majority of all its members, declare the place of any officer vacant, on his failure for one year, from inability or otherwise, to attend the Council meetings or perform the duties of his office. All vacancies shall be filled by the appointment of the Council, and any person so appointed shall hold office for the remainder of the term for which his predecessor was elected or appointed; *Provided*, that the said appointment shall not render him ineligible at the next annual meeting.

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Five members of the Council shall constitute a quorum; but the Council may appoint an Executive Committee, or business may be transacted at a regularly called meeting of the Council, at which less than a quorum is present, subject to the approval of a majority of the Council, subsequently given in writing to the Secretary, and recorded by him with the minutes.

V.

ELECTIONS.

The annual election shall be conducted as follows: Nominations may be sent in writing to the Secretary, accompanied with the names of the proposers, at any time not less than thirty days before the annual meeting; and the Secretary shall, not less than two weeks before the said meeting, mail to every member or associate (except honorary members), a list of all the nominations for each office so received, together with a copy of this rule, and the names of the persons ineligible for election to each office; and if the Council, or a Committee thereof, appointed for the purpose, shall have recommended any nominations, such recommendation may also be sent to members and associates with the said list of all nominations made, but not upon the same paper. And each member or associate, qualified to vote, may vote, either by striking from or adding to the names of the said list, leaving names not exceeding in number the officers to be elected, or by preparing a new list, signing said altered or prepared ballot with his name, and either mailing it to the Secretary or presenting it in person at the annual meeting; *Provided*, that no member or associate in arrears since the last annual meeting shall be allowed to vote until the said arrears shall have been paid. The ballots shall be received and examined by three Scrutineers, appointed at the annual meeting by the presiding officer; and the persons who shall have received the greatest number of votes for the several offices shall be declared elected, and the Scrutineers shall so report to the presiding officer. The ballots shall be destroyed, and a list of the elected officers, certified by the Scrutineers, shall be preserved by the Secretary.

VI.

MEETINGS.

The annual meeting of the Institute shall take place on the third Tuesday of February, at which a report of the proceedings of the Institute and an abstract of the accounts shall be furnished by the Council. Two other regular meetings of the Institute shall be held in each year, at such times and places as the Council shall select, and notice of all meetings shall be given by mail, or otherwise, to all members and associates, at least twenty days in advance. Special meetings may be called whenever the Council sees fit; and the Secretary shall call a special meeting on a requisition signed by fifteen or more members. The notices for special meetings shall state the business to be transacted, and no other shall be entertained.

Every question which shall come before any meeting of the Institute, shall be decided, unless otherwise provided by these Rules, by the votes of a majority of the members then present. Any member or associate may introduce a stranger to any meeting; but the latter shall not take part in the proceedings without the consent of the meeting.

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VII.

PAPERS.

The Council shall have power to decide on the propriety of communicating to the Institute any papers which may be received, and they shall be at liberty, when they think it desirable, to direct that any paper read before the Institute, shall be printed in the Transactions. Intimation, when practicable, shall be given, at each general meeting, of the subject of the paper or papers to be read, and of the questions for discussion at the next meeting. The reading of papers shall not be delayed beyond such hour as the presiding officer shall think proper; and the election of members or other business may be adjourned by the presiding officer, to permit the reading and discussion of papers.

The copyright of all papers communicated to, and accepted by, the Institute, shall be vested in it, unless otherwise agreed between the Council and the author. The author of each paper read before the Institute shall be entitled to twelve copies, if printed, for his own use, and shall have the right to order any number of copies at the cost of paper and printing, provided said copies are not intended for sale. The Institute is not, as a body, responsible for the statements of fact or opinion advanced in papers or discussions at its meetings, and it is understood that papers and discussions should not include matters relating to politics or purely to trade.

VIII.

AMENDMENTS.

These Rules may be amended at any annual meeting by a two-thirds vote of the members present; *Provided*, that written notice of the proposed amendment shall have been given at a previous meeting; *and Provided, also*, that the amendment or amendments so adopted shall be printed upon a ballot and sent, not later than the next distribution of printed matter, to all members and associates not in arrears for the preceding year (except honorary members and foreign members elected before February, 1880), and each person receiving the same shall be requested to return it to the Secretary with his written vote of Yes or No to each amendment, and his signature; and the President shall appoint as scrutineers three members or associates, who shall examine all of the said ballots which shall have been returned within one month from the date of their distribution, and shall report the result; and the Secretary shall publish and distribute to members, not later than the next distribution of printed matter, an announcement of the said result so reported, together with the text of the additional or amended rule or rules so adopted; and the amendment or amendments approved by the majority of the ballots so returned and reported shall become part of these rules from and after the publication of said announcement by the Secretary.

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PROCEEDINGS.

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PROCEEDINGS OF THE FIFTY-FOURTH MEETING, COLORADO, JUNE, 1889.

GENERAL COMMITTEE.

J. B. Grant, *Chairman;* E. W. Rollins, T. B. Stearns, *Secretaries;* P. H. van Diest, *Chairman of Sub-Committee on Papers;* G. E. Randolph, *Chairman of Sub-Committee on Transportation;* J. Milner, *Chairman of Sub-Committee on Hotels;* G. C. Munson, Hermann Beeger, E. S. Nettleton, J. K. Choate, N. P. Hill, H. B. Chamberlin, J. A. Cooper, F. Guiterman, John Evans, Regis Chauvenet, Frank Hall, C. H. Reynolds, H. E. Wood, B. H. Locke, L. H. Eicholtz, A. von Schulz, Henry Harrington, Edward Eddy, H. T. Rogers, W. H. James, John Pierce, A. S. Hughes, Wolfe Londoner, R. C. Hills, E. E. Burlingame, J. L. Routt, Aaron Gove, F. F. Chisolm.

LOCAL COMMITTEE AT PUEBLO.

A. Eilers, *Chairman;* A. S. Dwight, *Vice-Chairman;* John Dougherty, H. J. Fitch, E. R. Holden, Foster Nichols, W. W. Palmer, H. A. Keller, E. P. Mathewson, E. B. Kirby, P. B. de Schweinitz, J. W. Edwards, E. C. Eddie, W. H. Bunce, P. A. L. Mannheim, H. C. Mannheim, Reese James, W. C. Brace.

LOCAL COMMITTEE AT MANITOU.

William A. Bell, *Chairman.*

LOCAL COMMITTEE AT ASPEN.

Frederick G. Bulkley, *Chairman;* W. E. Newberry, J. C. Eames, D. W. Brunton, J. H. Devereux, H. B. Gillespie, D. R. C. Brown, J. M. Beach, W. A. Carlyle, G. W. Nyce, J. C. Carson, C. E. Doolittle, W. H. Yankee, A. W. Hare, A. F. Bardwell, W. J. Elmendorf, P. C. Gallagher, Henry Paul, C. E. Palmer, Geo. C. Hewett, H. K. Devereux, J. M. Downing, D. W. Strickland, W. B. Root, Lee Hayes, H. J. Mayhem, J. W. Deane, E. C. Babbitt, E D. Wright, W. J. H. Miller, H. E. Woodward, G. W. Lloyd, C. C. Morgan, Hal Sayr, Theodore Little, Jr., C. W. Miller, Frank Bulkley, W. R. Rust, E. F. Browne, H. D. Bates, F. M. Taylor, J. D. Hooper, G. W. Crowe, G. S. Newman, E. Gee, B. Ferris, F. M. Coombs, J. C. Bates, H. P. Cowenhoven.

LOCAL COMMITTEE AT LEADVILLE.

A. A. Blow, *Chairman;* A. V. Hunter, S. N. Dwight, Ned Steel, M. Londoner, J. Schloss, T. S. Wood, M. Boehmer, L. S. Noble, C. T. Limberg, J. H. Shotesbury, F. G. White, C. J. Carnahan, F. T. Freeland, J. W. Gaff, G. E. Taylor, J. H. Cragg, R. J. Coleman, W. R. Chadbourn, Jr., B. S. Buell, C. P. Schumacher, C. L. Hill, R. Estey, S. W. Mudd, S. F. Parrish, Dr. D. H. Dougan, C. C. Davis, H. H. Tomkins, R. J. Cary, F. Butler, C. Boettcher, Gen. F. M. Reardon, E. Smith, C. Sands, J. W. Smith, A. Sherwin, H. I. Higgins, F. Fohr, C. Hayden, Jr., Dr. B. S. Galloway, Dr. S. A. Bosanto, Dr. O. H. Simons, T. Jaycox, A. T. Thielen, W. W. Old, G. H. F. Meyer, G. W. Copeland, T. H. Oliver.

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The first session was held at 2 P.M., on Wednesday, June 19th, in the Assembly Room of the High School Building, Stout St., Denver. Hon. James B. Grant, Chairman of the General Committee, called the meeting to order and introduced Hon. Edward O. Wolcott, who extended to the Institute a cordial welcome to the State of Colorado. The hospitalities of the city of Denver were also tendered to the Institute by Mr. Wolfe Londoner, mayor of the city. President Pearce, with a brief allusion to the embarrassment of his position as both guest and host, requested the Secretary, as the representative of the visiting members, to reply to the addresses of welcome. At the close of the Secretary's response, President Pearce delivered the presidential address, embracing a general review of the progress of the metallurgy of gold and silver in the West during the past twenty-five years. (This address will be found on page 55 of this volume.)

The Secretary presented a report from the Council relating to the election of new members, and upon recommendation of the Council, the following persons were unanimously elected members and associates respectively. (The list includes also the names of those who were elected at subsequent sessions.)

MEMBERS.

Walter Hull Aldridge,	Pueblo, Colo.
Henry E. Armitage,	Lake City, Colo.
Alonzo F. Bardwell,	Aspen, Colo.
J. C. Bates,	Aspen, Colo.
Hiram P. Bellinger,	Pueblo, Colo.
J. A. Bradburn,	Syracuse, N. Y.
William H. Bradley,	Mingo Junction, Ohio.
Samuel Brady,	Detroit, Mich.
Edward F. Brown,	Aspen, Colo.
Frank Bulkley,	Aspen, Colo.
D. M. Campbell,	Birdsboro', Pa.
Charles F. Carnahan,	Leadville, Colo.
Franz Cazin,	Denver, Colo.
Hartwig A. Cohen,	Morey, Nev.
R. J. Coleman,	Leadville, Colo.
Frederick M. Coombs,	Aspen, Colo.
James P. Cooper,	Leadville, Colo.
Joseph Cozens,	Sault Ste. Marie, Ontario.
William C. Davis,	Denver, Colo.
James H. Devereux,	Aspen, Colo.
John C. Eames,	Aspen, Colo.
John Ellis Field,	Denver, Colo.
Alexander S. Garfield,	Boston, Mass.
Herman Garlichs,	Omaha, Neb.

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Henry B. Gillespie, ... Aspen, Colo.
Alvan P. Granger, ... Denver, Colo.
James B. Grant, ... Denver, Colo.
J. S. Greene, ... Denver, Colo.
Egbert B. C. Hambley, . . . Salisbury, N. C.
Hakon Hammer, ... Pottstown, Pa.
Strangman Hancock, ... Johannesburg, South Africa.
H. W. Hare, ... Aspen, Colo.
Albert Helms, ... Sydney, New South Wales.
Daniel Hillman, ... Birmingham, Ala.
A. B. W. Hodges, ... Pueblo, Colo.
H. B. Hodges, ... Omaha, Neb.
Charles A. Judkins, ... Leadville, Colo.
H. C. Lay, ... Telluride, Colo.
David E. Llewellyn, ... Powellton, W. Va.
George W. Lloyd, ... Aspen, Colo.
Franklin B. Locke, ... Detroit, Mich.
Albert P. Low, ... Ottawa, Ontario.
Jacob Lychenheim, ... Philadelphia, Pa.
Thomas Macfarlane, ... Ottawa, Ontario.
Mason W. Mather, ... Glenwood Springs, Colo.
Otto Metchke, ... Tucson, Arizona.
Louis G. Noble, ... Leadville, Colo.
R. H. Norton, ... Victoria, British Columbia.
Cortlandt E. Palmer, ... Aspen, Colo.
Henry Paul, ... Aspen, Colo.
J. D. Pennock, ... Syracuse, N. Y.
Edward A. Quintard, ... Batopilas, Mexico.
Morris Ramsey, ... Mt. Pleasant, Pa.
Robert Ramsey, ... Mt. Pleasant, Pa.
Otto Rissmann, ... St. Louis, Mo.
Edward W. Rollins, ... Denver, Colo.
Hal Sayr, ... Aspen, Colo.
M. E. Smith, ... Denver, Colo.
Charles Sooysmith, ... New York, N. Y.
John I. Souther, ... Hurley, Wis.
George A. Spotswood, ... Kingston, Ontario.
Thomas B. Stearns, ... Denver, Colo.
James H. Steedman, ... St. Louis, Mo.
John Storer, ... Sydney, New South Wales.
William P. Swarts, ... Chicago, Ill.
Cephas Taylor, ... Pittsburgh, Pa.
Horace P. Tobey, ... West Wareham, Mass.
Charles E. Willis, ... Chester Basin, Nova Scotia.
Tingley S. Wood, ... Leadville, Colo.
E. G. Woodford, ... Johannesburg, South Africa.
Henry E. Woodward, ... Aspen, Colo.
Edmond C. van Diest, ... San Luis, Colo.

ASSOCIATES.

Samuel L. Brown, ... Richard Center, Wis.

CHANGE OF STATUS.

Walter M. James, ... Philadelphia, Pa.

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The second session was held in the High School Building, on Wednesday evening.

In the absence of the author, a paper on The Concentration of Low-Grade Ores, by H. E. Armitage, Lake City, Colo., was read by P. H. van Diest.

The following papers were also read:

Concentration Before Amalgamation for Low-Grade, Partially Decomposed, Silver-Ores, with Notes on the Geology of the Flint Creek Mining District, by Chas. W. Goodale, Butte City, Montana and Wm. A. Akers, Phillipsburg, Montana.

The Ore-Deposits at Red Mountain, Ouray County, Colorado, by T. E. Schwarz, Red Mountain, Colo.

The third session was held in the High School Building on Thursday afternoon, June 20th, when the following paper was read and discussed:

The Geology and Ore-Deposits of Iron Mountain, Leadville, Colorado, by A. A. Blow, Leadville, Colo.

An informal session was held on Monday evening, June 24th, at the Wheeler Opera House, Aspen. Mr. Frederick G. Bulkley, Chairman of the Local Committee, explained the details of the programme that had been arranged for the Institute at Aspen, and invited the attention of members to an exhibition of specimens of the ores of that district.

The fourth regular session was held in the Wheeler Opera House on Tuesday morning, June 25th, Professor Thomas Egleston in the chair.

An address of welcome to Aspen was delivered by Hon. J. W. Deane, and acknowledged by the Secretary.

The following paper was read by W. E. Newberry, Aspen, Colo.:

Notes on the Geology of the Aspen Mining District.

The following papers were read by title:

An Occurrence of Copper Glance North of Lake Huron, with Notes on the Structure of the Locality, by J. T. B. Ives, Toronto, Canada.

An Improved French Pocket-Compass, by R. A. Bergier, Butte City, Montana.

The Desulphurization of Pyritiferous Iron-Ores, by Sterling G. Valentine, Lebanon, Pa.

Aluminum in Cast-Iron, by W. J. Keep, Detroit, Mich. (analytical determinations by Dr. C. F. Mabery, Cleveland, Ohio).

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The Dry Assay of Tin-Ores, by H. O. Hofman, Rapid City, Dakota.

Notes on the Manufacture of Open-Hearth Bridge-Steel, by N. W. Shed, Phœnixville, Pa.

The Distribution of Phosphorus in the Hudson River Carbonates, by Ingersoll Olmsted, Morristown, N. J.

The Influence of Silicon on the Determination of Phosphorus in Iron, by Thomas M. Drown, Boston, Mass.

The Coal-Trade and Miners' Wages in the United States in 1888, by Charles A. Ashburner, Pittsburgh, Pa.

Notes on the Additional Diaphragm in the Howell Roasting-Furnace, by Charles W. Goodale, Butte City, Montana.

Some Tests of the Relative Strength of Nitro glycerine and Other Explosives, by F. W. Clark, Chicago, Ill.

The Quartzite Gold-Deposits of Ouray District, Colorado, by G. E. Kedzie, Ouray, Colo.

The Iron Resources of Colorado, by Regis Chauvenet, Golden, Colo.

Avalanches, by B. E. Fernow, Washington, D. C.

The fifth and final session was held in the Wheeler Opera House on Tuesday evening, President Pearce in the chair. There was a general discussion of the paper presented at the morning session by Mr. W. E. Newberry, after which the following paper was read:

Imaginary Boundaries, by R. W. Raymond, New York City.

EXCURSIONS AND ENTERTAINMENTS.

Wednesday morning, June 19th, the members present were taken by special train to visit the smelting works of the Omaha and Grant Smelting and Refining Company, and of the Globe Smelting and Refining Company.

Thursday morning, June 20th, an excursion was made by special train to visit the smelting works of the Boston and Colorado Company at Argo.

Friday evening, June 21st, the members of the Institute were invited to a reception and lawn party on the grounds of the Colorado Smelting Company at Pueblo, where they were handsomely entertained by Mr. Eilers and other officers of the company.

Saturday, June 22d, was given to visiting, at Pueblo, the works of the Colorado Smelting Company, the Philadelphia Smelting and Refining Company, the Pueblo Smelting and Refining Company, and the Colorado Coal and Iron Company.

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In the afternoon the party was taken to Manitou, to remain until Monday morning. During the rest at Manitou, Dr. W. A. Bell opened his mansion, "Briarhurst," for two social receptions, and provided in many other ways for the entertainment of the visitors.

Monday, June 24th, was spent in going, via the Colorado Midland Railroad, from Manitou to Aspen. The latter place was reached too late for the meeting that had been arranged for by the Local Committee.

Tuesday, June 25th, visits were made between the morning and evening sessions to several of the mines on Aspen Mountain. Luncheon was served at the mines by the Local Committee.

The forenoon of Wednesday, June 26th, was devoted to the inspection of mines on Smuggler Mountain. After the return from the mines the members were taken to Hallam's Lake, where a bountiful dinner of freshly-caught trout was heartily enjoyed. Later in the day the party proceeded to Glenwood Springs. At this point opportunity was afforded to test the attractiveness and the luxury of the new monster swimming-bath, and in the evening an elaborate "complimentary lunch" was served at the Hotel Glenwood.

Thursday morning, June 27th, the party was taken by train to Newcastle to visit the coal-mines. At 1.30 P.M. the train left for Leadville, which place was reached about 6 o'clock. In the evening there was a brilliant reception, with dancing, at the City Hall, which was largely attended, both by the visiting members and by the citizens of Leadville.

On Friday, June 29th, the Local Committee provided transportation to such of the mines, mills, smelters and pleasure-resorts as the members wished to visit, and made ample provision for their hospitable reception and entertainment at all points of interest. The rooms of the Leadville Club were also thrown open for the use of members and their friends during their stay in the city. The courtesies so generously tendered were heartily accepted and warmly appreciated by the visiting members. In the evening the party left for Denver, where the members separated.

The place of holding sessions in Denver was put at the disposal of the Institute by the Colorado Scientific Society, which also extended an invitation to members to visit its museum and examine its collections.

MEMBERS, ASSOCIATES, AND GUESTS PRESENT.

The following names were registered at the Institute headquarters

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in Denver, but the list does not include all who took part in the meeting at places other than Denver.

Walter W. Allen.
Charles E. Anderson.
Henry E. Armitage.
Walter Atherton.
Hermann Beeger.
Francis C. Blake.
Henry W. C. Block.
A. A. Blow.
M. C. Bullock.
E. E. Burlingame.
Franz Cazin.
Samuel W. Cheyney.
Frederic F. Chisolm.
G. M. Davidson.
E. S. Davis.
William C. Davis.
P. B. de Schweinitz.
John M. Desloge.
W. F. Downs.
J. A. Haskell.
F. Aug. Heinze.
Charles E. Herbert.
G. C. Hewett.
H. B. Hodges.
Marmaduke B. Holt.
Philetus H. Holt.
Axel O. Ihlseng.
J. E. Johnson.
Aaron W. Kellogg.
Bradford H. Locke.
James McGinn.
Charles D. McLure.
Charles W. Miller.
E. S. Moffat, Jr.
Charles J. Moore.
William W. Morrison.
Walter T. Page.
Richard Pearce.
William H. Pettee.
Thomas Egleston.
W. H. Emanuel.
Francisco Escobar.
B. E. Fernow.
John E. Field.
Edouard L. Foucar.
Fred. Frazer.
F. T. Freeland.
Henry C. Freeman.
H. Van F. Furman.
Paul A. Fusz.
Herman Garlichs,
Charles W. Goodale.
Alvan P. Granger.
James B. Grant.
J. S. Greene.
H. M. Griffin.
Edward Griffith.
Walter C. Hadley.
S. H. Pitkin.
Robert E. Plumb.
Alfred Raymond.
R. W. Raymond.
Samuel Rider, Jr.
Lewis A Riley.
A. N. Rogers.
E. M. Rogers.
Edward G. Savage.
T. E. Schwarz.
P. W. Sheafer.
J. William Smith.
Edward G. Strober
Dolphus Torrey.
Edmond C. van Diest.
P. H. van Diest.
P. Van Dyk.
McLane Van Ingen.
William A. Wilson.

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PROCEEDINGS OF THE FIFTY-FIFTH MEETING, OTTAWA, ONT., OCTOBER, 1889.

LOCAL COMMITTEES.

Ottawa Executive Committee.—John Sweetland, M.D., *Chairman;* B. T. A. Bell, *Secretary;* Geo. Burn, *Treasurer;* Dr. A. R. C. Selwyn, C.M.G., Lieut.-Col. Anderson, C.E., Robt. Blackburn, C. Berkeley Powell, Robt. Bell, L.L.D., B.A.Sc., W. A. Allan, H. B. Small, E. D. Ingall, M.E., Hiram Robinson, J. Lainson-Wills, F.C.S., Chas. Magee (*Chairman of Finance*), Thos. Macfarlane, F.R.C.S., A. McLean.

Buckingham Committee.—S. P. Fanchot, *Chairman;* O. M. Harris, C. C. Hoyer Millar, F. S. Shirley, J. Lainson-Wills, A. Cameron, W. Mackintosh, T. W. Williams, R. C. Adams, Capt. Bothwell, R. Henwood, A. McNaughton, W. C. Mackenzie, A. Lomer, F. J. Wilson, Geo. Stewart, James McLaren, Alex. McLaren. W. A. Allan, Thos. Trimble, J. Keith Reid, W. J. Poupore, M.P.P., John Poupore, G. L. Parker, A. F. McIntyre, Q.C., Thos. Lyons.

Eastern Township Committee.—R. N. Hall, M.P., H. B. Ives, M.P., His Worship the Mayor of Sherbrooke, Dr. R. W. Ells, Lieut.-Col. Lucke, Dr. James Reed, T. P. Buck, Thos. Sheridan, Wm. King, G. Wertheim, John J. Penhale, M. Penhale, R. T. Hopper, J. N. Greenshields, Q.C., Col. Drew Gay, T. P. Bacon, L. J. Frechette, A. Johnson, A. Ward, W. Clearihue.

Sudbury Committee.—Dr. E. D. Peters, Jr., Duncan McIntyre, Geo. Attwood, M E., J. Merry, John Ferguson, F. L. Sperry, J. D. Evans, C.E.

Hotel Headquarters.—Russell House.

The first session was held on Tuesday evening, October 1st, in the Railway Committee Rooms, House of Commons. The meeting was called to order by Dr. John Sweetland, Chairman of the Local Committee, who called upon the Secretary of the Local Committee, B. T. A. Bell, to read letters of regret from T. Erratt, Mayor of Ottawa, Sir Hector Langevin and the Hon. A. S. Hardy. Dr. Sweetland, after a cordial welcome to the Institute, introduced the Right Hon. Sir John A. McDonald, whose witty and eloquent address was followed with the closest attention. The Hon. Mr. Ross spoke in behalf of the province of Quebec, while Alderman Henderson, of Ottawa, welcomed the visitors in behalf of the citizens. After a few remarks by Col. Richard Lay, United States Consul, Mr. John M. Garland, President of the Board of Trade of Ottawa, presented an address on behalf of that body. Prof. T. Egleston, briefly acknowledging this reception, called upon Dr. R. W. Raymond, who responded in behalf of the Institute. Prof. T.

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Egleston taking the chair, the proceedings were opened by the reading of the following papers:

Biographical Notice of George H. Cook, by J. C. Smock, Albany, N. Y.

Biographical Notice of William H. Scranton, by R. W. Raymond, New York City.

Biographical Notice of William Richard Jones, by Charles A. Ashburner, Pittsburgh, Pa. (Subsequently completed for publication, by R. W. Raymond, New York City.)

At the second session, held Wednesday afternoon, October 2d, the following papers were read:

The Mining Industries of Eastern Quebec, by Dr. Robert Ells, Ottawa, Ont.

Natural-Gas Explorations in the Eastern Ontario Peninsula, by Charles A. Ashburner, Pittsburgh, Pa.

At the third session, held on Thursday afternoon, October 3d, notice was given of the following amendment to the rules, proposed for adoption at the annual meeting:

It is proposed to amend Rule II., by inserting after the words "by ballot at a regular meeting," in each of the two places where these words occur in the Rule, the words:

"Or by ballot through the mail, at the discretion of the Council."

The following papers were read:

The Geological Relations of the Principal Nova Scotia Minerals, by E. Gilpin, Jr., Halifax, N. S.

Notes on the Port Arthur Silver District, by E. D. Ingall, Ottawa, Ont.

Gold Mining in Nova Scotia, by John E. Hardman, Oldham, N. S.

The treatment of Fine Gold in the Sands of Snake River, Idaho, by Prof. T. Egleston, New York City.

The following papers were read by title:

Gold-Quartz, by W. M. Courtis, Detroit, Mich.

Notes on Some Coals in Western Canada, by W. Hamilton Merritt, Toronto.

The Wear of Rails as Related to their Section, by P. H. Dudley, New York City.

Phosphorus in Cast-Iron, by W. J. Keep, Detroit, Mich.

The Davis-Colby Ore-Roaster, by S. G. Valentine, Lebanon, Pa.

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A Proposed Method for Working Tellurides, by Frank Clemes Smith, Ann Arbor, Mich.

Notes on the Republic of Colombia, S. A., by John C. F. Randolph, New York City.

The Form of Crater Produced by Exploding Gunpowder in a Homogeneous Solid, by Frank Firmstone, Easton, Pa.

The Occurrence and Treatment of the Argentiferous Manganese-Ores of Tombstone District, Arizona (additional to the former paper, *Trans.*, xvii., 767), by Charles W. Goodale, Butte, Mont.

The fourth and final session was held on Thursday evening, October 3d. The following paper was read:

The Sudbury Ore Deposits, by Dr. E. D. Peters, Jr., Sudbury, Ont.

Dr. Robert Bell, Ottawa, Ont., gave an oral account of the phosphate-deposits of Ottawa county, some of which were to be visited on the following day, and the following paper in discussion of the subject was read by title:

List of Commercial Phosphates, by W. H. Adams, New York City.

The following resolution, proposed by Mr. John C. F. Randolph, was unanimously adopted:

Resolved, That the Secretary be instructed to express to the Local Committee the congratulations of the Institute upon their complete and efficient arrangements for the Ottawa meeting; and to the governments, corporations and citizens of Canada, who have so generously contributed to the pleasure, comfort and profit of the visiting members and guests of the Institute, the hearty acknowledgements and thanks of those who have been privileged to take part in the sessions and excursions of this meeting.

Resolved, That the Council be requested to present to the Library of the Parliament of the Dominion of Canada, in recognition of the extraordinary courtesies officially extended to the Institute, and also to Sir John A. MacDonald, whose cordial kindness, together with that of Lady MacDonald, has so greatly enhanced the pleasure of the members of the Institute, a complete bound set of the *Transactions* of the Institute.

The following persons, having been duly recommended by the Council, were unanimously elected:

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LIST OF MEMBERS.

O. P. Ankeny,	Clinton, Iowa.
Rawlinson Tennant Bayliss,	Marysville, Mont.
Charles S. Bedell,	New York, N. Y.
David A. Bennett,	Bessemer, Mich.
H. P. Butz,	Uniontown, Pa.
Sidney Bretherton,	New York, N. Y.
Frank D. Carney,	Thomaston, Maine.
Robert L. Crooke,	New York City.
G. W. Crowe,	Aspen, Colo.
T. R. Countryman,	Denver, Colo.
Robert N. Dickman,	Cleveland, O.
James Douglas,	New York, N. Y.
Abram T. Eastwick,	Philadelphia, Pa.
William H. Echols, Jr.,	Rolla, Mo.
Martin L. Griffin,	Holyoke, Mass.
George R. Gray,	Johannesburg, S. Africa.
James Otis Handy,	Pittsburgh, Pa.
Arthur Hodge,	La Salada, U. S. Colombia.
James P. Hosie,	Scranton, Pa.
George E. Jenkins,	Dover, N. J.
Ralph A. Jones,	Newberry, Mich.
Guy Roche Johnson,	Longdale, Va.
Charles Kaufman,	St. Louis, Mo.
Landon Ketchum,	S. Norwalk, Conn.
Wilbur A. Langdon,	Allentown, Pa.
William A. Leonard,	West Wareham, Mass.
Charles H. Livingstone,	Denver, Colo.
Bernard MacDonald,	Tombstone, Ariz.
Ferdinand McCann,	Pueblo, Colo.
Samuel D. Mills,	St. Ignace, Mich.
Charles McCrery,	Troy, N. Y.
August R. Meyer,	Kansas City, Mo.
J. N. R. Molson,	Quebec, Canada.
Buck E. Ogden,	Philadelphia, Pa.
J. Obalski,	Quebec, Canada.
R. A. F. Penrose, Jr.,	Little Rock, Ark.
J. C. Peters,	Helena, Mont.
James S. Pomeroy,	Oswego, Ore.
G. S. Ramsay,	Pueblo, Colo.
John M. Reid,	Oxford Gold Mines, Nova Scotia.
J. Schumacher,	Butte City, Mont.
Henry B. Seaman,	Philadelphia, Pa.
Boyd M. Smith,	Tichbone, Ont., Canada.
Hy. Beaumont Small,	Ottawa, Canada.
John Prentice Terry,	El Paso, Tex.
Enrique Touceda,	Troy, N. Y.
Edgar G. Tuttle,	Eagle Pass, Tex.
William Viggers,	Durango, Mexico.
Harrison L. Wadsworth,	Denver, Colo.

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E. A. Wetmore,	Marquette, Mich.
W. H. Yankee,	Aspen, Colo.
Frederick A. Yeager,	Uniontown, Pa.

ASSOCIATES.

Theron R. Gue,	Halifax, Nova Scotia.
Thomas Sharp,	Nashville, Tenn.
James W. H. Stubbs,	Wimbledon, England.
J. S. Walker,	Birmingham, Ala.
Benjamin C. Watson,	Huantla, Estado Morelos, Mexico.
Benjamin C. Wilson,	Waverly, Nova Scotia.

CHANGE OF STATUS.

Karl Eilers,	Brooklyn, N. Y.
A. S. J. Mahony,	New York, N. Y.
John Stanton,	New York, N. Y.

Excursions and Entertainments.

Wednesday morning was occupied by a drive to the Chaudiere Falls, an inspection of mills and factories, and a visit to the Central Experimental Farm. In the evening there was an informal social gathering at the Russell House.

On Thursday morning, visits were made to the Museum of the Geological Survey of Canada, the Parliamentary Library, the Senate and House of Commons, the Fisheries Museum, the Art Gallery and the works of the Canadian Granite Company.

Friday was given over to an excursion to the phosphate-mines of Ottawa County, the members being conveyed by the Canadian Pacific Railway to Buckingham, whence a steamer carried them to Little Rapids Landing, where the party was entertained at luncheon. Returning to Buckingham, an informal reception was tendered at the Crystal Rink, Ottawa being reached late in the evening.

On Saturday morning, the greater number of the members started on an excursion through the Eastern Townships and the Province of Quebec, spending Saturday and Sunday at Montreal, where they were hospitably entertained.

Reaching Sherbrooke on Monday noon, a special train carried the party to the asbestos mines of the Black Lake and Thetford districts, where the rest of the day was spent. The quarries of the Dudsville Lime and Marble Company at Marblehead were visited on Tuesday morning, the remainder of the day being passed at Sherbrooke, where the Mayor and Corporation tendered a public

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reception. On Wednesday, Richmond was reached, a visit being paid to the new Rockland slate quarries.

A second excursion left Ottawa on Saturday, October 5th, for Sudbury, Ont., where the mines and works of the Canadian Copper Company, the Dominion Mineral Company and other mines were visited.

A contemplated excursion to the Port Arthur silver-mining district was not carried out.

A handsome souvenir, in the form of an album of photographs of scenes in the vicinity of Ottawa, was presented by the Local Committee to all the visiting members and their ladies.

Members, Associates and Guests Present.

R. C. Adams.
S. N. Anderson.
Charles A. Ashburner.
S. E. Bretherton.
H. T. Bovey.
J. H. Bramwell.
C. S. Bedell.
B. T. A. Bell.
Dr. Robert Bell.
John Birkinbine.
W. B. Cogswell.
J. B. Church.
W. S. DeCamp.
C. A. DeCamp.
T. Egleston.
R. W. Ells.
Martin L. Griffen.
T. R. Gue.
W. H. Hulick.
J. E. Hardman.
L. Holbrook.
W. H. Hale.
B. J. Harrington.
W. H. Hutcheson.
C. H. Jouët.
James E. Jopling.
James T. B. Ives.
Elfric Drew Ingall.
C. Kirchhoff, Jr.
J. S. Lane.
N. M. Langdon.
R. G. Leckie.
A. P. Lowe.
C. MacDowell.
Thomas Macfarlane.
A. W. Newell.
J. Obalski.
Jos. C. Platt.
F. Prince.
Ed. D. Peters, Jr.
A. C. Rand.
John C. F. Randolph.
R. W. Raymond.
Theodore D. Rand.
F. R. Redpath.
I. M. Reis.
J. C. Smock.
S. R. Sharples.
T. Tumble.
Jos. F. Wells.

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PROCEEDINGS OF THE FIFTY-SIXTH (TWENTIETH ANNUAL) MEETING, WASHINGTON, D. C., FEBRUARY, 1890.

Local Committees.

General Committee.—General W. S. Rosecrans, *Chairman;* F. P. Dewey, *Secretary.*

Executive Committee.—S. F. Emmons, *Chairman;* Comdr. F. M. Barber, Dr. T. M. Chatard, Dr. D. T. Day, F. P. Dewey, Arnold Hague.

Banquet Committee.—S. F. Emmons, *Chairman;* Andrew Carnegie, Fred. H. Newell.

Halls and Reception Committee.—Dr. T. D. Day, *Chairman;* Maj. J. W. Powell, Josiah Pierce, W. A. Raborg.

Excursion Committee.—Comdr. F. M. Barber, *Chairman;* M. B. Kerr, E. Richards, Dr. F. P. McLean, E. O. Leech, Dr. W. Kempster.

Badges and Printing Committee.—Arnold Hague, *Chairman;* George H. Eldridge, H. Hollerith, E. W. Parker.

Finance Committee.—Dr. T. M. Chatard, *Chairman;* P. E. Chapin, G. L. Bradley B. E. Fernow, C. E. Coffin.

Hotel Headquarters.—The Arlington.

The opening session was held on Tuesday evening, February 18th, at the United States National Museum. The meeting was called to order by General W. S. Rosecrans, chairman of the Local Committee of Arrangements, who welcomed the members to Washington. Addresses of welcome were also made by Prof. S. P. Langley, Secretary of the Smithsonian Institution, and Major J. W. Powell, Director of the United States Geological Survey, to which President Pearce appropriately responded.

The following papers were read by the Secretary in the absence of the authors:

Biographical Notice of Charles Albert Ashburner, by Prof. J. P. Lesley, Philadelphia, Pa.

Biographical Notice of Franklin B. Gowen, by Eckley B. Coxe, Drifton, Pa.

Dr. Raymond supplemented Mr. Coxe's paper with personal reminiscences of Mr. Gowen.

The President appointed as scrutineers, to examine the ballots for officers of the Institute, Prof. R. H. Richards, Dr. C. B. Dudley and Dr. T. M. Drown.

After the adjournment of the session, opportunity was given for an informal social reunion, and for the inspection of the collections in the National Museum.

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The second session was held at the Arlington hotel on Wednesday evening, February 19th. At the invitation of the Council, Prof. T. C. Mendenhall, Superintendent of the United States Coast and Geodetic Survey, gave an interesting account of the new international standards of length and mass, and exhibited fac-similes of the standard meter and kilogramme recently deposited with the Coast Survey. (See page 716 of this volume.)

The following papers were read:

Crystalline Magnetite in the Port Henry, New York, Mines, by John Birkinbine, Philadelphia, Pa.

Electricity and Haulage, by F. A. Pocock, Scranton, Pa.

President Pearce then read the presidential address, on The Association of Gold with other Metals in the West. (See page 447 in this volume.)

The following paper was read by the author, and illustrated with lantern-views:

Stripping Ore-Deposits, by F. H. McDowell, New York City.

The third session was held at the Arlington on Friday morning, February 21st, when the following papers were read:

The Properties of Aluminum, with Some Information Relating to the Metal, by Alfred E. Hunt, John W. Langley, and Charles N. Hall, Pittsburgh, Pa.

Aluminum in Search of a Nickname, by Oberlin Smith, Bridgeton, N. J.

Aluminum in the Drawing-Press, by Oberlin Smith, Bridgeton, N. J.

Note on the Use of Aluminum in the Construction of Instruments of Precision, by Prof. W. P. Blake, New Haven, Conn.

The Secretary then read the

Annual Report of the Council.

In accordance with the Rules, the Council makes the following report to the Institute:

The financial statement of the Secretary and Treasurer, duly audited, shows receipts for the year from all sources, including the balance at the beginning of the year, of $29,223.99, and expenditures of $18,714.11, leaving a surplus of $10,509.88. In addition to this, the Treasurer holds United States bonds, belonging to the Institute, as per last annual report, of the par value of $7300, making the total amount of cash on hand and securities at par $17,809.88.

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Since the balance from the last statement was $5880.28, the actual receipts of the year were greater by $4629.60 than the expenditures.

The surplus above stated includes the fund composed of the dues paid by life members, of whom there are now 68.

The detailed statement is as follows:

Statement of the Secretary and Treasurer, of Receipts and Disbursements, from February 1st, 1889, to January 31st, 1890.

Balance from statement, February 1st, 1889,		$5,880.28
Received for dues from members and associates,	$18,187.41	
" life memberships,	1,300.00	
" sale of publications,	1,642.09	
" binding *Transactions*,	969.14	
" authors' pamphlets,	408.80	
" " engraving and electrotyping,	164.86	
		22,672.30
" interest on United States bonds and deposits,		671.41
		$29,223.99
Paid for printing volume xvii. of *Transactions*,	$2,214 24	
" " pamphlet editions of papers,	1,569.85	
" " authors' editions of papers,	411.75	
" " list of members,	152.50	
" " mailing list,	73.25	
" " circulars and ballots,	233.41	
" binding volume xvii. of *Transactions*,	942.78	
" " other volumes of *Transactions*,	306.60	
" " exchanges,	86.06	
" engraving and electrotyping,	1,224.90	
" postage, including P. O. Box rent,	673.10	
" incidental expenses of mailing pamphlets,	45.92	
" stationery, including post-paid envelopes,	392.60	
" rent of offices,	800.00	
" janitor's fees,	102.00	
" express and freight charges,	54 1.25	
" telegrams,	31.45	
" proof-reading, volume xvii. and index,	100.00	
" insurance,	9.00	
" coal, stove and repairs,	59.67	
" small expenditures, unclassified,	88.23	
" salaries of secretary, assistants and typewriters,	7,933.20	
" expenses of secretary,	580.84	
" storage on *Transactions*, advanced for 1890,	138.51	
		$18,714.11
Balance,		10,509.88
		$29,223.99

The statement made in the last annual report concerning the value placed upon the back-volumes of the *Transactions* is confirmed by

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the foregoing figures, which show that $1642.09 was received during the year from this source. This total includes also the amount paid for copies of single papers in pamphlet form, which constitutes, however, but a trifling part of the whole. There is still a supply of complete sets of the *Transactions;* but members desiring to own all the volumes would do well to purchase such as they lack before certain numbers (more particularly Volumes IX., X. and XVII.) are exhausted.

Three meetings have been held during the year: the nineteenth annual meeting (being the fifty-third of the Institute) at New York, in February; a June meeting at Denver and Aspen, Colorado, and an October meeting at Ottawa, Canada. They were as pleasant and as profitable as the meetings of the Institute have always been—what more could be said? Those who have enjoyed the privilege of attending our periodical reunions need no further proof of their peculiar charm, while for those who have not been enlightened by this experience, any other evidence could not take its place.

As is usually the case in years when meetings are held in the far West, the publication of Volume XVII. of the *Transactions* was somewhat later than usual; and Volume XVIII. is likely to be similarly delayed unless the authors of papers will give more prompt co-operation to the efforts of the Secretary to hasten publication.

Changes in membership have taken place during the year as follows: 205 members and 20 associates have been elected; 11 associates have become members; 11 members have been reinstated on payment of arrears; 1 honorary member, 11 members and 1 associate have died; 29 members and 6 associates have resigned; and 44 members and 6 associates have been dropped from the list for nonpayment of dues.

These changes are tabulated as follows:

		H. M.	F. M.	M.	A.	Totals.
	At date of last report	11	45	1606	168	1830
Gains	by Election	.....	.....	205	20	225
	" Change of Status	.....	.....	11	.....	11
	" Reinstatement	.....	.....	11	.....	11
Losses	" Resignation	.....	.....	29	6	35
	" Dropping	.....	.....	44	6	50
	" Change of Status	.....	.....	.....	11	11
	" Deaths	1	.....	11	1	13
Total gains		.....	.....	227	20	247
" losses		1	.....	84	24	109
Present membership		10	45	1749	164	1968

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The list of deaths comprises the names of John Percy, honorary member; C. A. Ashburner, Edward Bailey, Joshua E. Clayton, George H. Cook, F. S. Craven, H. S. Craven, Daniel Hillman, W. R. Jones, J. J. O'Farrell, W. H. Scranton, N. S. Stockwell, members; and F. B. Gowen, associate.

An interesting event of the year was the excursion to Great Britain and the continent of Europe, organized primarily by the American Society of Mechanical Engineers, and joined individually by many of the members of the Institute. An account of this trip, prepared for the Council by Mr. A. E. Hunt, a Vice-President, and Mr. Charles Kirchhoff, Jr., a Manager of the Institute, will be published as an appendix to this report, in order to place on record in the *Transactions* a suitable recognition of the distinguished and abundant courtesy extended by our professional brethren in other countries to the representatives of American engineering.

Since the return of the visiting party, a copy of the following address, beautifully engraved and illuminated, has been sent to the Council of the Institute:

To the Presidents of the American Societies of Civil, Mining, Mechanical and Electrical Engineers: We, the President, Past Presidents, Vice-Presidents, Council, and members of the Institution of Civil Engineers, acting on this occasion both for ourselves and the various bodies of engineers of the United Kingdom, hereby tender to you, as representatives of the members of the several engineering societies of America, a sincere and cordial welcome to this country, and gladly avail ourselves of the opportunity of publicly acknowledging, and as far as possible reciprocating, the manifold courtesies which for many years past have been lavished on British engineers visiting the Great Republic.

It is a source of peculiar satisfaction to receive you within this building, because we are here in the home of the parent of all the duly constituted engineering societies of this kingdom.

The association of engineers in England into one body, for their common advantage, originated with Smeaton, one of the fathers of the profession; it was not, however, until twenty-five years after his death, viz., in 1817—well-nigh three-quarters of a century ago—that the present Institution was actually formed.

Telford became its President in 1820, and in 1828 it received the Royal Charter, under which it has ever since flourished. In regard to magnitude, it is sufficient to state that our institution now comprises (including the class of students) upwards of 5700 members, and is largely adding to its numbers every year.

Although the civil engineers act as hosts in your reception, the several engineering bodies of the country are associated with us; and others outside our profession join in the welcome, and have rendered valuable aid in our endeavors to secure your comfort and gratification.

Foremost of all must be mentioned the special permission given by Her Most Gracious Majesty the Queen, for you to visit and inspect her royal palaces and domains at Windsor and in the metropolis. Nor must we omit to place on record the very exceptional and gratifying fact that the Lord Mayor, aldermen and Com-

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mon Council of the City of London have been pleased to place at the disposal of the Reception Committee the use of their ancient and noble Guildhall in order that we may entertain you, in accordance with old English custom at a festival dinner therein.

The leading railway, dock, gas and water companies have vied with each other in exercising hospitality, nor have private individuals been lacking in the earnest desire to add to your gratification whilst in our midst.

It would be superfluous and presumptuous to enlarge on the professional merits of American engineers. Their great works and clever inventions have passed far beyond the sphere of mere technical appreciation, and have become of world-wide celebrity. We feel justified in regarding the influence of you and your predecessors as one of the principal factors which have raised, with unexampled rapidity, the modest republic of George Washington to one of the foremost nations on earth.

The problem of dealing with great difficulties presented by Nature, and, until recently, with comparatively limited means and limited appliances, has been solved by the American engineer, and the solution has left its mark upon the character of the nation.

With a population about double our own, and a territory stretching between ocean and ocean, more than 3000 miles from east to west, not to speak of its extent from north to south, distances have been conquered by your vast system of railways on a scale of magnitude of which we have no experience.

We trust that all the arrangements made for your visits to some of the most important public works in this kingdom will be successful and acceptable, and we hope you may carry back pleasant recollections of your visit to this country.

Witness our hands and seal at Westminster this 13th day of June, 1889.

JOHN COODE, *President.*
JAMES FORREST, *Secretary.*

This graceful compliment has been suitably acknowledged on behalf of the Institute; but the debt of thanks still owing to our foreign brethren cannot be paid in words. It is with great pleasure, therefore, that the Council looks forward to an early opportunity of co-operating in a more substantial recognition and return of hospitalities. This opportunity is furnished by the acceptance, on the part of the Iron and Steel Institute of Great Britain, of the invitation extended by this Council, to hold its annual meeting this year in the United States. While this Institute has thus taken the formal initiative, it has been cordially supported by sister societies, and by the American iron and steel trade generally; and the arrangements for receiving and entertaining our distinguished guests will be made by a general committee, representing all interests and associations concerned. It is sufficient to say at this time, that the Council has decided to hold an autumn meeting of this Institute in connection with the meeting of the British society—probably immediately afterwards; and that, in order to give ample time and leisure for all preparations, it has been decided to hold no spring or summer meet-

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ing of the Institute this year. Members are earnestly requested to prepare papers for the autumn meeting and send them to the Council for acceptance and publication early. It is highly desirable that the papers of that meeting should be printed beforehand, to insure profitable discussions.

The Council will bring before the Institute at this annual meeting for discussion, modification and adoption, the amendment to Rule II., notice of which was originally given at the Boston meeting, in February, 1888, and again at the Buffalo meeting, October, 1888. At the following annual meeting, held in New York city, in February, 1889, this amendment was not acted upon, and consequently went over as unfinished business. As proposed, it reads as follows:

Amend Rule II. by inserting after the words "by ballot at a regular meeting." in the seventh and eighth lines of the rule, and also after the same words in the eleventh line of the rule, the following parenthesis:

"(or by ballot at any time conducted through the mail, as the Council may prescribe)."

The purpose of this amendment is to authorize the Council, in its discretion, to substitute for the present method of electing members, the more deliberate and dignified method of a general ballot by mail. This will save the time now spent at meetings of the Institute in reading lists of candidates and formally electing them. This waste of time would be intolerably increased if a ballot were taken in literal compliance with the rule. On the other hand, the alternative, which has become our traditional practice, of instructing the Secretary to cast the ballot for the Institute, is of questionable propriety, and could be at any time made impracticable by the objection of a single member present.

The proposed amendment will not deprive the Council of the discretionary power to submit the names of candidates for membership to be elected by vote of the members present at a meeting of the Institute. This may still be deemed desirable in special cases; or, if experience with the ballot by mail should be unsatisfactory, the Council can return to the old system until a better one shall have been devised.

In operating experimentally the new method proposed, the Council will make trial of the following procedure:

1. The names of persons proposed, concerning whom information may be desired, will be sent by circular to all members, and the Council will thus be assisted in its judgment by the confidential opinion and recommendation of the members at large.

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2. After a sufficient number of names shall have been approved by the Council, they can be then sent out upon a membership ballot for election by mail. Candidates will be relieved from the necessity of waiting until a meeting of the Institute occurs, at which they may be elected. The election of members will go on independently of the meetings. There will be no urgent reason that names, presented, perhaps, during the sessions of a meeting, should be immediately acted upon, lest the candidates be condemned to wait for months. It is believed that, in many ways, this system may constitute an improvement, satisfactory to all concerned.

APPENDIX.

THE EUROPEAN TRIP OF THE JOINT PARTY OF ENGINEERS IN 1889.

THE first suggestion of a visit of members of American engineering societies to Europe was made at a dinner given in London by the President of the Institution of Mechanical Engineers of Great Britain to two members of the American Society of Mechanical Engineers, to which subsequently an official invitation was sent. The final outgrowth of the movement was, that the Institution of Civil Engineers of Great Britain, the older and more comprehensive body, assumed the leadership in England, and that members of the American Institute of Mining Engineers and of the American Society of Civil Engineers joined the party, of whom 170 sailed from New York on May 25th, in the "City of Richmond," specially chartered, while 105 left on the 29th in the "City of New York." Of the total number, seventy-one were members of the American Institute of Mining Engineers.

Both vessels anchored in the Mersey at Liverpool, the "City of Richmond" on the afternoon of June 4th, and the "City of New York" on June 6th, in the early morning.

The engineers upon both vessels were welcomed by a sub-committee of the Liverpool Reception Committee, headed by Mr. Alfred Holt, Chairman, and Chairman of the Liverpool Dock Board, as well as the largest individual ship-owner in the world. Mr. Daglish and Mr. West also were members of the sub-committee.

On Wednesday, June 5th, the party arriving by the earlier steamer divided into two groups, most of the ladies with a few of the gentlemen going to the Knowsley Hall, the county-seat of the Earl of Derby, the major portion of the gentlemen being taken to the Liverpool end of the Manchester Ship Canal works.

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On Thursday, after the arrival of the "City of New York," the Executive Committee of the Joint European Excursion of the American Engineering Societies in 1889 was appointed as follows:

Honorary Chairman.—D. J. Whittemore, Past Pres. A. S. C. E.

Chairman.—Henry R. Towne, Pres. A. S. M. E., member A. S. C. E.

Honorary Secretary.—C. E. Emery, member A. S. C. E., member A. S. M. E., member A. I. M. E.

Secretary in England and Germany.—Charles Kirchhoff, Jr., Mgr. A. I. M. E., member A. S. M. E.

Secretary in France.—G. M. Bond, Mgr. A. S. M. E., member A. S. C. E.

Treasurer.—Alfred E. Hunt, Vice-Pres. A. I. M. E., member A. S. C. E., member A. S. M. E.

Other Members of Committee.—O. Chanute, Past Pres. A. S. C. E., member A. I. M. E.; C. J. H. Woodbury, Vice-Pres. A. S. M. E., member A. S. C. E.; Thos. C. Clarke, member A. S. C. E., member A. I. M. E.; Prof. F. R. Hutton, Sec'y A. S. M. E., member A. I. M. E.; Wm. H. Wiley, Treas. A. S. M. E., member A. S. C. E., member A. I. M. E.; A. Dempster, member A. S. C. E.; Wm. Kent, Vice-Pres. A. S. M. E., member A. I. M. E.; James Archbald, member A. S. C. E.; S. W. Baldwin, Mgr. A. S. M. E.; Clark Fisher, member A. S. C. E., member A. I. M. E.; J. T. Hawkins, Mgr. A. S. M. E.; Dr. Herbert G. Torrey, member A. I. M. E.; Wm. Forsyth, Mgr. A. S. M. E., member A. I. M. E.; Oberlin Smith, member A. S. M. E., member A. I. M. E.; E. V. d'Invilliers, member A. I. M. E.

Later in the day of Thursday, June 6th, two alternate excursions were offered to the visitors, one being to the Mersey Docks and Harbor Estate, the other to Messrs. Laird & Brothers' extensive Birkenhead Iron Works. The excursion upon the steamer Lancashire around the Mersey Docks was honored by the unusual courtesy that the shipping was dressed in flowing bunting and gala attire. Most of the party returned via the Mersey Tunnel Railroad from the Birkenhead Docks to Liverpool. The large party who visited the Messrs. Laird's works were received by the Mayor of Birkenhead at the Town Hall. Thence they were conveyed to the private residence of Mr. Wm. Laird, where they were entertained. After examining the extensive works of the Messrs. Laird, and the hydraulic lift arrangements and engines of the Hamilton Square station, the pumping and ventilating station of the Mersey Tunnel, they

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returned through the Mersey Tunnel to Liverpool. In the evening a conversazione was held by the Mayor, Mr. E. H. Cookson, in the Town Hall. A very complete map of Liverpool and Birkenhead, showing the Docks and Harbor Estate, was especially prepared for the visitors, and was bound up in a pamphlet programme for distribution.

The Reception Committee at Liverpool consisted of the President of the Institution of Civil Engineers, Sir John Coode, Sir Frederick Bramwell, Sir Lowthian Bell, Sir Jas. Allport, Mr. Adamson, Sir Henry Bessemer, Sir George Bruce, Mr. Cowper, and many others, as well as the efficient and hospitable Secretary, Mr. James Forrest.

On Friday, June 7th, the party separated for the interval of the Whitsuntide holidays, the major portion of the party taking one of three itineraries offered by the London & North Western R. R., at reduced rates; the first tour going via Manchester through North Wales and to the Menai Straits, visiting Stephenson's Tubular and Telford's Suspension Bridges, and via Crewe to London. The second party visited the English lake-district, beginning at Barrow-in Furness, and then proceeding via Kenilworth, Warwick, Leamington and Stratford-on-Avon to London. The third party went via Manchester, Crewe, Chester, Birmingham, Stratford-on-Avon and Oxford to London.

Before starting on the tours referred to, a large portion of the entire party visited the Crewe works of the L. and N. W. R. R. on Friday, and availed themselves of the kind invitation of Mr. F. W. Webb, mechanical engineer. A portion of the party were as cordially and hospitably entertained the same day, visiting the new railway-shops of the Lancashire and Yorkshire Railway, as the guests of Mr. J. F. Aspinwall. A complimentary banquet given by the Manchester Reception Committee of the Institution of Civil Engineers was preceded by a reception by the Mayor of Manchester, assisted by the Mayors of Salford, Oldham and Stockport. Saturday, June 8th, a special train conveyed the greater number of this party along the line of the extensive excavations of the Manchester ship canal, from the Manchester end viewing the site of the various docks, and Brindley's famous aqueduct. The party were conducted over the works by Mr. W. Leader-Williams, the engineer-in-chief of the ship canal. During the morning the party were guests of Mr. T. A. Walker, the contractor of the canal. Each guest was provided during the trip, by the in-

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defatigable Mr. Thomas Ashbury, the honorable secretary of the reception committee, with a long list of the notable mills which have made Manchester world-famous, to which a cordial invitation had been given the American engineers, whom Mr. Ashbury personally undertook to conduct at any time during their visit that any of the party desired.

On Thursday, June 13th, commenced the eight days of hospitality in London, with a special choral service held in Westminster Abbey, followed by an interesting address by Dean Bradley in the chapel of Henry VII., on the sacred and historical associations of the Abbey. At noon a visit was made by special arrangement to the Houses of Parliament. In the afternoon a formal reception was tendered by the Institution of Civil Engineers at No. 25 Great George Street, at which there was presented to the visiting engineers an address, beautifully engrossed and illuminated, from the Institution. [Illuminated copies of this address were subsequently forwarded to the American societies concerned; and the text is quoted in the Annual Report of the Council.] The presentation speech was made by the president, Sir John Coode, K. C. M. G. Prof. Robert H. Thurston, a past vice-president of the American Institute of Mining Engineers, made a suitable response.

In the evening the most striking courtesy of our visit was extended in a dinner, given by the Institution of Civil Engineers of Great Britain, at the noble Guildhall of the city of London, by the express sanction of the Lord Mayor, Aldermen, and Court of Common Council of the city of London. Only once before had a similar courtesy ever been extended to any other than a municipal party visiting London. Addresses were made by Hon. Robert T. Lincoln, Minister to England; Sheriff Newton, Sir John Coode, Mr. D. J. Whittemore, Henry R. Towne, Alfred E. Hunt, Elihu Thompson, and Thomas C. Clarke.

Among the guests of the occasion were the American Minister, Mr. Robert T. Lincoln, Sir Edward Thornton Lord Armstrong, Arch-deacon Farrar, Dean Bradley, Sir Henry Bessemer, Sir William Thompson, Mr. Latimer Clarke, Sir James Douglass, and many others whose names are familiar to Americans.

On Friday, June 14th, the party was divided into three groups. The first visited the London Docks and the Beckton Gas Works, and the Metropolitan Main Drainage and Sewage Purification Works at Barking.

The second party left the stairs at the Westminster bridge and

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went to the foundation and piers of the Tower bridge in process of erection; thence to the Marine Engine Works of Humphreys, Tennant & Co., at Deptford Pier, and to a whitebait-luncheon at the celebrated Ship Inn at Greenwich. They also visited the Naval Museum and the Painted Hall of the Greenwich Hospital, and the shipbuilding and torpedo vessel yard of the Yarrow Brothers at the Isle of Dogs. After an inspection of the Dock and Freight Plant of the North London Railway at Poplar the party returned to London.

The third party went to the Marine Engine Works of J. & G. Rennie, Blackfriars; thence to a plant of the London Hydraulic Power Company, and the works of Peter Brotherhood, Maudslay, Field & Co., the Hydraulic Company, the Metropolitan Main Pumping Station, and afterwards to Lambeth Palace. In the chapel of the Palace the party were received by Dr. Benson, the Archbishop of Canterbury, who addressed them in fitting and kindly words. From here the engineers and ladies went to the Doulton Potteries, and through the art-studios which are a part of the factory.

On Saturday, June 15th, a special train conveyed the party to Windsor Castle and its interesting and historic grounds. By consent of Her Majesty, many rooms in the building, which are not open to the public, were made accessible to the visitors. After the visit to the chapels, state apartments, mews and towers, luncheon was served in the hall of the Albert Institution in Windsor town, The Mayor and other civic officers were present at the President's table. After luncheon the party were taken in open carriages through the Windsor Great Park and the Long Walk. Others of the party proceeded to the residence of Mrs. S. B. Boulton, where "Mid-Summer Night's Dream" was most enjoyably performed upon the grounds, as a summer theatrical entertainment.

In the evening Lord Brassey gave a reception at his private house.

On Monday morning, June 17th, by the Queen's permission, the party visited St. James's Palace, Buckingham Palace, and the Royal Mews, in London. At St. James's Palace, the time of the visit was so arranged that the party were there at the guard-mount, and a complimentary concert was given by the band, American national airs being played during the ceremony. At Buckingham Palace, also, the party were allowed access to the rooms which are only entered by the Queen's own subjects on the occasions of drawing-rooms or similar social or privileged occasions.

In the afternoon the Baroness Burdett-Coutts gave a reception.

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On Monday evening, an official meeting of the party was held in the theatre of the Institution of Civil Engineers.

On Tuesday, there were two alternate excursions; one going by special train, furnished by the London and Southwestern Railroad, to Hampton Court, thence by barge on the Thames to the Water Works, thence down through the lock to Hampton Court Palace, and, after a visit to the galleries, gardens, and grounds, including, as elsewhere, parts of the palace premises usually closed to public visitors, the party returned either by special train or through the courtesy of one of our hosts, by fast launches down the Thames to Richmond, and so home by train.

The other party went to Wimbledon to inspect the sewage disposal works of the local board at that point; thence to the engine works of Willens & Robinson. Afterwards this group joined the others, and returned either by train or by Messrs. Robinsons' launches with the other party, by a most delightful sail in the evening moonlight down the Thames river.

On Wednesday, June 19th, there were three alternate excursions; first, to visit the freight warehouse of the London and Northwestern Railroad, or similar visits to the Midland Railway Co.'s works, or to the iron tunnelling works of the London and Suffolk Sub-way. Thence the three parties went together to the London Electric Supply Corporation's Works, at Deptford, and in the afternoon to the flower show of the Royal Botanical Society in Regent's Park. In addition to these, smaller parties were arranged to accept private invitations; notable among them, was that of Professor Tyndall, who delightfully entertained a party of twelve at Hindhead, Haslemere.

Others were very hospitably received by Mr. Pope, the Treasurer of the Middle Temple, who carried the party over that most interesting series of buildings and entertained them at luncheon. Professor Bauerman also conducted a party interested in mineralogy over the British Museum and the Geological Museum in Jermyn Street.

Besides this, Sir Henry Bessemer was in attendance at Messrs. Ford & Wright's Diamond Cutting and Polishing Works.

During their stay in London, the visitors were made honorary members of the Whitehall Club in Westminster. Invitations were also received to be present at the conversazione given at South Kensington by the Society of Arts on the week following that upon which they left for Paris.

Sir John Fowler, K.C.M.G., and Mr. Baker, engineer of the Forth Bridge, gave cordial invitations to visit the structure.

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On Thursday morning, June 20th, a special train left Victoria station to convey the party to Dover, on its way to Paris. The party was accompanied by Sir John Coode, President of the Institution of Civil Engineers, and James Forrest, the Secretary.

Upon the arrival at Calais, the party were welcomed by MM. A. Brüll and Gottschalk, past presidents of the Society of Civil Engineers of France, and by several of the vice-presidents and leading members of that society, among them being M. Alphaud, M. Jousselin, M. Banderali, M. Garnier, M. Charton, and M. Haton de la Goupillière; Mr. Octave Chanute and Mr. Pontzen, members of the American Society of Civil Engineers, who had preceded the party to Paris, and who had, with Mr. James Dredge, done much to arrange for its reception, were also present.

Between Calais and Paris a stop was made at Arques, to inspect the hydraulic canal-lift.

On Saturday morning, the 22d of June, at the headquarters of the Civil Engineers of France, on the first floor of the Machinery Palace at the Exposition, the members of the party were presented to Mr. G. Eiffel, President of the French Society; and after an address of welcome by Mr. Eiffel and a reply by Mr. Chanute, were escorted to the base of the Eiffel tower, and were taken in successive lifts to the upper platform. Upon the return to the first stage a luncheon was served in the restaurant Brébant.

In the afternoon, members of the French Society divided themselves into groups, representing respectively Mining, Metallurgy, Machinery and Spinning, Iron and Copper and Sugar Works, Electricity, and Public Works, to conduct the members through the Exposition.

On Monday morning, June 24th, two parties went through the wonderful sewers of Paris. Others visited the Gobelin Tapestry Works and the hospital of M. Pasteur. The fourth group had the entree to the Paris Observatory, to Pasteur's Hospital, and to the École des Mines. In the afternoon visits were planned to the works of the General Company of Paris Cabs, their laboratory and stations, and to the General Omnibus Company. There was also an excursion this afternoon to the compressed-air works of the Parisian Compressed Air Co., employing the system of M. Popp. At the close of this latter professional visit, an entertainment was given to a considerable portion of the party in the Bois de Boulogne.

On Tuesday, June 25th, there were two visits in the morning, the first to the special points of interest in the Exposition, and the sec-

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ond to the sewage-works of Gennevilliers and the sewage-farm. In the afternoon, the Historical Museum of the City of Paris, the Conservatoire des Arts et Métiers, the Hotel des Invalides, and the Jardin des Plantes, were visited.

On Wednesday June 26th, visits were made to the Hotel de Ville, the Hotel de Cluny, the National Library, and the National Manufactory of Sèvres china. In the evening, a general meeting of the party was held at the headquarters of the French Civil Engineers.

On Thursday, June 27th, M. Paul Decauville, Senior, invited the party to visit the railroad works at Petitbourg; and in the evening a favored few, by invitation, occupied the private box at the Grand Opera which is set aside for the President of the Republic.

President Carnot also gave an audience to a select deputation at his palace in the Champs Elysées, and a similar attention was shown by the Prefect of the Seine. There was also an address delivered to the party in the City Hall by representative gentlemen selected from the French House of Duputies.

On Saturday, June 29th, the Minister of Public Works of France, M. Yves Guyot, gave a ball at which a considerable number of the visiting engineers were present.

On Sunday, June 30th, many of the engineers were invited, with their lady friends, to a musicale at the house of President Carnot, where a reception was held by President and Madame Carnot.

On Monday, July 1st, the party broke up; many going home via England; many starting for independent trips in different parts of Europe, while a large number, especially of the American Institute of Mining Engineers, accepted the invitation conveyed through Mr. E. Schroedter to be the guests of the Coal and Iron Masters of Westphalia, the Rhenish provinces, and Luxembourg.

On Tuesday morning, July 2d, the party visited the Rothe Erde Works, and the Zinc and Lead Works at Stolberg, while the ladies were taken in charge by Mr. Parsons, the American Consul at Aix La Chapelle. Later in the day the party reached Düsseldorf.

On Wednesday morning, July 3d, the party inspected the double shaft of the Colliery Zollverein; and later under the guidance of Theodore Möller and Hüssener, the coal distillation of Bulmke; and the works of the Schalker Gruben und Hütten Verein, under the guidance of Mr. F. Burgers, the General Manager.

After the return to Düsseldorf, a conversazione was offered in the Zoological Garden by the Nieder Rheinische Bezirksverein Deutscher Ingenieure. A welcome was given to the party by Mr. R. M. Daelen,

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to which Mr. Oberlin Smith, member of the American Institute of Mining Engineers, and now president of the Society of Mechanical Engineers, responded. After a supper given to the visitors, they enjoyed the general illumination of the garden in their honor, and a ball was given them later in the evening.

On Thursday, July 4th, a party left for the Rhine Steel Works, Meiderich, near Ruhrort. Others visited the forges, rolling-mills, foundries, cast-steel works, coke ovens, coal mines, furnaces, and gas-engines, of the neighborhood, at least twenty-five different establishments sending cordial invitations to the visiting engineers.

On both days, while the gentlemen of the party were engaged in examining works of professional interest, the ladies visited the famous art galleries of Düsseldorf, and some of the studios.

On the evening of the 4th of July, a complimentary banquet was given by the iron masters and colliery proprietors to their American guests.

On Friday, the party went to Cologne, visited the great cathedral, and then to Coblenz, where the party was received by Herr Karl Spaeter, an official of the city, and conveyed to the castle of the late Empress Augusta, the widow of the late William I. A deputation, consisting of Messrs. Oberlin Smith, Walter Wood and Charles Kirchhoff Jr., were affably received by the Empress, who addressed them in English. The party were thereafter conducted through the castle and private gardens, into which the public are not allowed to enter. By direction of the Empress, a collation was spread in the garden and the entire party partook of the hospitality of the Empress of Germany.

After this visit, they were taken to the wine cellars of Messrs. Deinhardt & Co. Driving thence to the Rhine, a special chartered steamer carried the party down to Koenigswinter. Thence by rack railroad, they were conveyed to the Drachenfels, where again a banquet was served. Returning to the steamer, the party sailed down the Rhine to Cologne. On the way, salutes were interchanged between the steamer and the various residences and towns on the banks of the river, and in many cases illuminations and fire-works were given; at Cologne, the walls of the city and the boat bridge were illuminated, thousands of people having assembled.

The party scattered here, at the close of this day's entertainment, most of them going back to Great Britain, and from thence home. Many of the party assembled on July 24th at the shops of the Midland Railroad, where they were hospitably entertained by Mr.

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John Noble, the Chairman, and several directors of the Midland Railway. Most of the party returned home in small detachments, at their convenience, during the months of July, August and September.

On the 20th of July, a most enjoyable reunion was held, in the form of a small dinner .given by Mr. James Dredge in London, at which there were presented to Mr. James Forrest, and to Mr. James Dredge, handsomely-chased and inscribed loving-cups, by the American Engineers, as a recognition of all that had been done for them during their stay in London.

The amendment to Rule II., stated and recommended in the report of the Council, was then unanimously adopted, subject to the vote of the Institute by mail, as provided in Rule VIII.

The Scrutineers appointed at the first session submitted their report declaring the following officers to have been elected:

PRESIDENT.

ABRAM S. HEWITT, ...	New York City.

VICE-PRESIDENTS.
(To serve for two years.)

HENRY M. HOWE, ...	Boston, Mass.
S. F. EMMONS, ...	Washington, D. C.
H. S. MUNROE, ...	New York City.

MANAGERS.
(To serve for three years.)

JOSEPH D. WEEKS, ...	Pittsburgh, Pa.
W. J. TAYLOR, ...	Philadelphia, Pa.
S. T. WELLMAN, ...	Cleveland, O.

TREASURER.

THEODORE D. RAND, ...	Philadelphia, Pa.

SECRETARY.

ROSSITER W. RAYMOND, ...	New York City.

The fourth and final session was held in the hall of the Cosmos Club on Friday afternoon, February 21st.

The Secretary read, in abstract, the following papers:

Aluminum and other Metals Compared, by W. J. Keep, Detroit, Mich.

Aluminum in Wrought Iron and Steel Crstings, by W. J. Keep, Detroit, Mich.

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The following papers were read by the authors:

Aluminum Bronze and Brass as Suitable Materials for Propellers, by Eugene H. Cowles, Lockport, N. Y.

The Heroult Process of Smelting Aluminum Alloys, by F. P. Dewey, Washington, D. C.

Notes on the Gold-Deposits of Montgomery County, Maryland, by S. F. Emmons, Washington, D. C.

The Lewis and Bartlett Bag-Process of Collecting Lead-Fumes at the Lone Elm Works, Joplin, Missouri, by F. P. Dewey, Washington, D. C.

The following papers were read by title:

Notes on the Geology of the DeKaap, Gold-Fields in the Transvaal, by W. H. Furlonge, Barberton, South Africa.

Phosphorus in Pig-Iron, Steel and Iron-Ores, by Clemens Jones, Hokendauqua, Pa.

Electrical Accumulators or Storage-Batteries, by Pedro G. Salom, Philadelphia, Pa.

The Rights of the Owner or Possessor of a Lode Mining-Claim, by Carl Henrich, Webb City, Mo.

Railway Splice-Bars, and Specifications for their Manufacture, by Robert W. Hunt, Chicago, Ill.

Physical Properties of Some of the Alloys of Manganese, Copper, and Aluminum, by Eugene H. Cowles, Lockport, N. Y.

A System of Rail-Sections in Series, by P. H. Dudley, New York City.

Uintaite, Albertite, Grahamite and Asphaltum Described and Compared, with Observations on Bitumen and Its Compounds, by William P. Blake, New Haven, Conn.

Wurtzilite from the Uintah Mountains, Utah, by William P. Blake, New Haven, Conn.

Notes on Fuel-Gas, by George W. Goetz, Milwaukee, Wis.

Peculiar Working of a Blast-Furnace, by N. B. Wittman, Birdsboro, Pa.

Filling and Blowing-In at the Durham Blast-Furnace, by B. F. Fackenthal, Jr., Riegelsville, Pa.

Notes on American Cannel Coal, by Graham Macfarlane, Louisville, Ky.

Geology of the Choctaw Coal-Field, by H. M. Chance, Philadelphia, Pa.

Note on Gold-Mining and Milling in Korea, by Willard Ide Pierce, New York, N. Y.

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The Rush Creek, Arkansas, Zinc-District, by H. M. Chance, Philadelphia, Pa.

Note on the Friction of Mine-Car Wheels, by R. van A. Norris, Wilkesbarre, Pa.

The Velocity of Bodies of Different Specific Gravity, Falling in Water, by Robert H. Richards and A. E. Woodward, Boston, Mass.

Preliminary Note on the Thermal Properties of Slags, by H. M. Howe, Boston, Mass.

Specifications for Cast-Iron Coated Water-Pipe, by Thomas W. Yardley, Chicago, Ill.

Notes on the Energy and Utilization of Fuel—Solid, Liquid and Gaseous, by W. J. Taylor, Philadelphia, Pa.

The following persons, duly recommended by the Council, were unanimously elected as members and associates:

Honorary Members.

Prof. Moritz Gaetzschmann,	Freiberg, Saxony.
Bergrath Adolph Paters,	Vienna, Austria.
Bergrath Albert Serlo,	Berlin, Germany.

Members.

Randolph Adams,	Emmaville, New South Wales.
George S. Andrus,	Telluride, Colo.
James Wm. Ashcroft,	Mount Costigan, New South Wales.
R. A. Ayers,	Estillville, Va.
Henry C. Allen,	Yonkers, N. Y.
John F. Allen,	Buenos Ayres.
C. L. Beckwith,	Bisbee, Ariz.
T. A. Bennet,	London, England.
T. Elton Bott,	Nottingham, England.
Horace E. Burt,	Black River Falls, Wis.
C. G. Buchanan,	New York City.
John H. Bartlett,	Lynchburg, Va.
Charles Jefferson Clark,	St. Louis, Mo.
Romaine Clarke Cole,	Waterbury, Conn.
Edward D. Chester,	London, England.
T. F. Cole,	Negaunee, Mich.
Clarence A. De Camp,	Boonton, N. J.
W. H. Doane,	Cincinnati, Ohio.
Bertram H. Dunshee,	Butte City, Mont.
Alfred J. Dunstan,	Mount Costigan, New South Wales.
L. G. Eakins,	Washington, D. C.
John P. Edwards,	Uniontown, Pa.
H. Le Neve Foster,	West Hartlepool, England.

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Rufus Foster, Jr.,	Scranton, Pa.
Frank Eugene Fielding,	Virginia City, Nev.
Charles L. Flack,	Newark, N. J.
Henry Froehling,	Richmond, Va.
Franklin Farrel,	Ansonia, Conn.
Charles Martin Hall,	Pittsburgh, Pa.
Wm. Huntley Hampton,	Portland, Oregon.
James Kendall Harvey,	Redruth, Cornwall.
Edwin N. Hawkins,	Denver, Colo.
William D. Hofius,	Leetonia, Ohio.
John Wesley Howell,	Bisbee, Ariz.
W. A. Hudson,	Florence, Ala.
Thomas A. Ireland,	Denver, Colo.
William Kelly,	Vulcan, Mich.
Holbert S. Kerr,	Salt Lake City, Utah.
Mark B. Kerr,	Washington, D. C.
Carl W. Kolbe,	Midland, Mo.
Lionel Redmond Lenox,	S. Bethlehem, Pa.
Robert Packer Linderman,	S. Bethlehem, Pa.
John W. Langley,	Pittsburgh, Pa.
Daniel Webster Langdon, Jr.,	Cincinnati, Ohio.
J. Harry Lee,	Baltimore, Md.
Ambrose A. McHose,	Norristown, Pa.
Charles W. Mead,	Pony, Mont.
Charles E. Mills,	Bisbee, Ariz.
J. H. MacNaughton,	Iron Mountain, Mich.
Harvey Bartlett Nichols,	Leadville, Colo.
F. Osmond,	Paris, France.
James Park,	Auckland, New Zealand.
Arthur L. Pearse,	Lima, Peru.
John A. Potter,	Munhall, Pa.
Charles Page Perin,	Londonderry, Nova Scotia.
Abraham S. Patterson,	Philadelphia, Pa.
Theodore Cuyler Patterson,	Philadelphia, Pa.
Lester L. Robinson,	San Francisco, Cal.
Sanford Robinson,	San Francisco, Cal.
F. B. Richards,	Cleveland, Ohio.
W. C. Ralston,	San Francisco, Cal.
David Davey Rosewarne,	Adelaide, S. Australia.
M. K. Salsbury,	Pittsburgh, Pa.
J. Uno Sebenius,	Mineville, N. Y.
Robert James Scott,	Pittsburgh, Pa.
Frederick H. Smith,	Baltimore, Md.
Alfred Samuel Godden Smith,	Sheet Harbor, Nova Scotia.
Stephenson Percy Smith,	New Zealand.
Henry Lloyd Smyth,	Boston, Mass.
Thomas Shields,	Allegheny City, Pa.
W. H. Seamon,	Rolla, Mo.
Joseph J. Taylor,	Sabinal, Mexico.
William A. Thacher,	Tegucigalpa, C. A.
Gray Torrey,	Stirling, N. J.

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James B. Tonking,	Mount Hope, N. J.
George Henry Frederick Ulrich,	Dunedin, New Zealand.
Claude Vautin,	London, England.
Arthur Harvey Wethey,	Butte City, Mont.
Thomas Weir,	Granite, Mont.
Dr. Otto Wuth,	Pittsburgh, Pa.
John W. Worthington,	Birmingham, Ala.
Paul Wilkes,	Charlotte, N. C.
Charles H. Wellman,	Cleveland, Ohio.
Lawrence Hersee Whitmore,	London, England.
John F. Wilkinson,	New Almaden, Cal.

ASSOCIATES.

W. A. Beddoe,	New York City.
John Brunnings,	New York City.
L. V. Brown,	Jackson, Ohio.
Henry Darlington,	Pittsburgh, Pa.
John H. Jones,	Philadelphia, Pa.
Edward O. Leech,	Washington, D. C.
Thomas D. Ledyard,	Toronto, Canada.
J. Langcloth,	New York City.
James McKeen,	Easton, Pa.
Arthur B. Meeker,	Washington, D. C.
Park Painter,	Pittsburgh, Pa.
Levi Quier,	Reading, Pa.
William Scallon,	Butte City, Mont.
A. P. Smith,	Washington, D. C.
Ferdinand Schlesinger,	Milwaukee, Wis.

The following Associates were, by unanimous vote, upon the recommendation of the Council, made Members:

Robert G. Brown,	Butte City, Mont.
W. S. Maharg,	Bethlehem, Pa.
Thomas Sharp,	Nashville, Tenn.
Clement Stewart,	Easton, Pa.
J. S. Walker,	Birmingham, Ala.

Upon motion of Mr. E. C. Pechin, the Secretary of the Institute was directed to express the very hearty thanks of the Institute to the local Committee of Arrangements, and to others in Washington, who had done so much to make the visit of the Institute delightful, both socially and professionally.

The President of the Institute, after congratulating the members on the success of the sessions, and the pleasure of the excursions and entertainments, declared the meeting adjourned.

EXCURSIONS AND ENTERTAINMENTS.

On Wednesday, February 19th, a delightful excursion was made

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to Mount Vernon by a large number of members and guests.

On Thursday morning, February 20th, an interesting visit was made to the United States Navy Yard. In the afternoon, various points of attraction in and around Washington were visited by small parties.

On Thursday evening, more than 150 members and guests took part in a very pleasant subscription dinner at the Arlington.

At 4.30 P.M., on Friday, February 21st, a large number of members and guests were received at the Executive mansion by the President and Mrs. Harrison.

During Friday evening, the visiting members with their ladies and friends were cordially received and entertained at the residences of Maj. J. W. Powell and Mr. S. F. Emmons, meeting at both places many distinguished representatives of the social and scientific circles of Washington.

On Saturday morning, a party of about 50 was conveyed in omnibusses and carriages to the newly opened gold-mining district of Montgomery County, Maryland, described in the paper read by Mr. Emmons, at one of the sessions of the day before. The party returned to the Arlington about 5 P.M., after an outing full of pleasure and interest. On the way to the mines the great arch at Cabin John bridge was inspected.

By the courtesy of the managers of the Cosmos Club, the privileges of the Club were enjoyed during the week of the meeting by the members of the Institute.

MEMBERS, ASSOCIATES AND GUESTS PRESENT.

The following persons were registered at the headquarters of the Local Committee:

Acker, E. O'C.,	Bethlehem, Pa.
Adams, William H.,	New York, N. Y.
Atkins, William,	Pottsville, Pa.
Ayres, W. S.,	McCainsville, N. J.
Ayres, Mrs. W. S.,	McCainsville, N. J.
Bachman, F. E.,	Salem, Va.
Barber, F. M.,	Washington, D. C.
Beck, William H.,	Washington, D. C.
Birkinbine, John,	Philadelphia, Pa.
Birkinbine, Miss,	Philadelphia, Pa.
Blair, Andrew A.,	Philadelphia, Pa.
Blake, Francis C.,	Mansfield Valley, Pa.
Blake, William P.,	New Haven, Conn.
Boies, H. M.,	Scranton, Pa.
Boies, Mrs. H. M.,	Scranton, Pa.

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Boyer, Jerome L.,	Reading, Pa.
Braeunlich, Mrs.,	New York, N. Y.
Brown, Mrs. N. A.,	New York, N. Y.
Carson, James P.,	Dobbs Ferry, N. Y.
Pechin, Mrs. E. C.,	Cleveland, O.
Pechin, Miss M. S.,	Cleveland, O.
Pechin, Miss B. S.	Cleveland, O.
Pennock, J. D.,	Syracuse, N. Y.
Pierce, J. S.,	Washington, D. C.
Pierce, Willard Ide,	New York, N. Y.
Pocock, Francis A.,	Scranton, Pa.
Pocock, Mrs. Francis A.,	Scranton, Pa.
Porter, J. C.,	Pittsburgh, Pa.
Porter, Mrs. J. C.,	Pittsburgh, Pa.
Powell, J. W.,	Washington, D. C.
Prentiss, Mrs. F. H.,	New York, N. Y.
Price, Thomas,	San Francisco, Cal.
Price, Miss O. T.,	San Francisco, Cal.
Price, Miss Minnie E.,	San Francisco, Cal.
Raborg, W. A.,	Washington, D. C.
Raymond, R. W.,	Brooklyn, N. Y.
Raymond, Mrs. R. W.,	Brooklyn, N. Y.
Raymond, Miss Elizabeth,	Brooklyn, N. Y.
Rich, Jacob M.,	New York, N. Y.
Richards, E.,	Washington, D. C.
Richards, Robert H.,	Boston, Mass.
Richards, Mrs. Ellen H.,	Boston, Mass.
Riordan, D. M.,	Flagstaff, Ariz.
Riordan, Mrs. D. M.,	Flagstaff, Ariz.
Roberts, Percival, Jr.,	Philadelphia, Pa.
Robertson, Kenneth,	Goshen, Va.
Roscrans, W. S.,	Washington, D. C.
Rothwell, R. P.,	New York, N. Y.
Rothwell, Miss,	New York, N. Y.
Scaife, W. L.,	Pittsburgh, Pa.
Sharples, S. P.,	Boston, Mass.
Smith, Boyd,	Washington, D. C.
Smith, A. P.,	Washington, D. C.
Smith, A. D. W.,	Pottsville, Pa.
Smith, M. E.,	Denver, Col.
Smith, Oberlin,	Bridgeton, N. J.
Smith, Mrs. Oberlin,	Bridgeton, N. J.
Smith, Miss Emily,	Washington, D. C.
Spaulding, Hollen C.,	Boston, Mass.
Spilsbury, E. Gybbon,	Trenton, N. J.
Strobel, Victor O.,	Philadelphia, Pa.
Szemelenyi, Miss,	Washington, D. C.
Taylor, Charles,	New York, N. Y.
Taylor, W. J.,	Philadelphia, Pa.
Thaw, W., Jr.,	Pittsburgh, Pa.
Thaw, Mrs. W., Jr.,	Pittsburgh, Pa.
Thompson, Robert M.,	New York, N. Y.
Torrey, Herbert G.,	New York, N. Y.
Torrey, Mrs. Herbert G.,	New York, N. Y.
Torrey, Gray,	New York, N. Y.

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Torrey, Dolphus,	New York, N. Y.
Torrey, Miss Nellie,	New York, N. Y.
Torrey, Miss Julia,	New York, N. Y.
Weeks, Joseph D.,	Pittsburgh, Pa.
Weeks, Mrs. Joseph D.,	Pittsburgh, Pa.
Wellman, S. T.,	Cleveland, O.
Wellman, Mrs. S. T.,	Cleveland, O.
Wellman, Miss B.,	Cleveland, O.
Wheelock, Jerome,	Worcester, Mass.
Whitehorne, Mrs. E. C.,	New York, N. Y.
Wilcox, John F.,	Pittsburgh, Pa.
Wilkes, J. Frank,	Charlotte, N. C.
Wilkes, John,	Charlotte, N. C.
Williams, David,	New York, N. Y.
Williams, Mrs. David,	New York, N. Y.
Williams, R. R.,	New York, N. Y.
Williams, Mrs. R. R.,	New York, N. Y.
Williams, Miss,	New York, N. Y.
Williams, H.,	Butte City, Mont.
Wister, Jones,	Philadelphia, Pa.
Wolfe, Albert H.,	Chicago, Ill.
Wyatt, Francis,	New York, N. Y.
Wyatt, Mrs. Francis,	New York, N. Y.
Wiley, William H.,	New York, N. Y.
Zehnder, C. H.,	Berwick, Pa.
Zehnder, Mrs. C. H.,	Berwick, Pa.

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PAPERS.

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*THE DRY ASSAY OF TIN-ORES.**

BY H. O. HOFMAN, RAPID CITY, DAKOTA.

(Colorado Meeting, June, 1889.)

THE tin-deposits of the Black Hills, discovered some ten years since, are beginning to attract very general attention, and the question how to treat the ores to the best advantage has become a pressing one. No systematic investigation of this subject had been made until quite recently, when it was taken up by the Dakota School of Mines, and the ores tested as to their tenor and proper mode of treatment. In the assay laboratory numerous determinations were made of the percentage of black tin in the ore-bodies, and of the metallic tin contained in it. The occurrence of tin in the Black Hills being different from that in other parts of the world, the usual mode of procedure had to be somewhat modified.

The object of this paper is to give the results of the writer's effort to discover the best method of assaying the tin-ores in the dry way. His task divided itself under three heads, in the following order:

1. To test the different methods of assaying purified black tin in the dry way.

2. To study the influence upon the assay of the different minerals associated with the black tin.

3. To find by what simple and rapid means the cassiterite could most effectually be freed from its gangue.

The apparatus used was that to be found in any ordinary assay laboratory (the pulp-balance being sensitive to 1 mgr.), as it was the aim to furnish quick and reliable methods with simple appliances. That dry assays of black tin have not the same degree of accuracy as wet assays is well known, but they suffice for the determinations necessary in metallurgical work. If great exactness is required, recourse must be had to one of the two wet methods, that of Rose

* NOTE BY THE SECRETARY.—This paper was prepared for presentation also as a thesis for the degree of Ph.D. at Ohio University (Athens, O.). It is published, however, in the *Transactions* of the Institute only.

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(fusing with sulphur and sodium carbonate, and converting the insoluble stannic oxide into soluble stannic sulphide) or that of Rivot (reducing the stannic oxide in a current of hydrogen to metallic tin, which is then readily soluble).

The cassiterite occurring in the Black Hills tin-ores is quite pure, as is shown by the following analyses* of lode-tin and stream-tin:

Lode-tin:	Occidental Mine, ...	96.42 SnO_2
	Tin Mountain, ...	97.50 "
	First Find, ...	94.70 "
Stream-tin:	Nigger Hill, ...	92.60 "
	" "	93.00 "
	Southern Hills, ...	92.80 "

Stannite has lately been found in one locality. The cassiterite has an adamantine luster and its color varies from brown to black, the light varieties being rare. The minerals associated with the cassiterite that are of interest to the assayer are iron pyrite, arsenopyrite, in a very few instances galena and blende, quartz, feldspar, mica, tourmaline, garnet, and columbite (tantalite). The sulphides occur rarely, and then always in small quantities; quartz and silicates form the gangue; columbite (tantalite), although sometimes found in the same veins, can always be separated by hand-sorting. When it has accumulated in the stream-tin deposits, which occasionally happens, it remains with the purified black tin. Bismuth, copper, and tungsten, which occur frequently in the deposits of other countries, are found very sparingly.

I. The Assay in General.

Tin that is to be assayed in the dry way must be present in the form of oxide. The assay proper consists of applying in a crucible at a high temperature reducing and fluxing reagents, which bring the stannic oxide to the metallic state and convert the gangue into a liquid slag. The behavior of stannic oxide and of metallic tin at an elevated temperature causes, however, certain difficulties, which make the results often less accurate than could be desired. These difficulties are:

First, that some of the resulting metallic tin is liable to be volatilized.

Second, that metallic tin decomposes the alkali carbonates used as fluxes, forming stannates, which enter the slag.

* *Transactions*, xvii., pp. 595, 596.

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Third, that at the temperature required to reduce the stannic oxide, other metallic oxides become also reduced to the metallic state and alloy with the resulting tin button, and vitiate the assay.

Fourth, that sulphates are reduced, or, if sulphides are present, they combine directly with the metallic tin and carry it into the slag.

Fifth, that the gangue in the ore is always siliceous. This causes loss, as the stannic oxide, while passing through the intermediate state of stannous oxide, combines with silica and silicates.

Sixth, that stannic oxide is liable, before its reduction has begun, to combine with the basic fluxes used in assaying and form stannates; this liability increases as the quantity of the flux is increased.

Thus it will be seen that the results can be too high (as when iron enters the tin button) or too low (as when tin is carried off in the slag as stannate, sulphide, or silicate), and that all impurities should be removed, if possible, before the actual dry assay takes place. The making up of the charge must also be regulated to counteract as much as possible the bad influences of the basic fluxes.

The crucibles used in assaying tin-ores are the ordinary clay crucibles, the plumbago crucible, and the porcelain crucible. The naked clay crucible has the disadvantage that the basic fluxes attack the clay and form silicates, which are liable to slag some of the tin. From the results obtained by the writer it would seem that the material of the Battersea crucibles used by him must have been much better than that of the crucibles referred to by Berthier,* as no appreciable loss resulted from their use. P. Ricketts† recommends the lining of the crucible with chalk. This is probably to avoid the action of the basic fluxes on the acid material of the crucible, as the calcium oxide, being a strong base, protects the clay of the crucible from the corrosive action of the alkali, and thus hinders the formation of an alkali silicate. An outline of his method having been obtained from Dr. Ricketts, the lining was made from the so-called "prepared chalk" of the druggist in the following manner: The prepared chalk was mixed with water in an enameled iron dish to the consistency of cream. It was then allowed to stand awhile, that the coarse, gritty particles might settle and collect at the bottom. The chalk-emulsion, now freed from impurities, was poured off and was ready for use. To line the crucible, this was half filled with the chalk-mixture, the upper half being coated by giving the

* *Traité des Essais par la Voie Sèche*, Liége, 1847, vol. ii., p. 485.

† *Notes on Assaying*, 8th ed., New York, 1886, pp. 88, 89.

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crucible a rotary motion, after which the excess of the liquid was poured off. The crucible was then placed bottom up for five minutes to drain, after which it stood for two hours to dry, and was then ready for use. The coating obtained was thin, uniform, and smooth. If the crucible be filled entirely with the mixture, and this be allowed to remain a little while before being poured off, the coating will be less uniform; on the sides, part of the naked crucible will be exposed, and the lining at the bottom, being too thick, will crack when it dries. If a greater thickness is wanted than can be obtained by one coat, it can only be properly put on a layer at a time, as above, until the desired thickness is reached. The effect of the lining was disappointing; it did not, as will appear later, prove useful, but rather the reverse.

Another highly recommended lining is one of charcoal. The fluxes do not act chemically upon it, and its reducing power makes it unnecessary to add any charcoal to the assay; a reducing atmosphere is kept up during the entire operation. Berthier* states that he obtained 4 per cent. more tin in a brasqued crucible than in a naked one although even then he recovered 2 per cent. less tin than was present.

The following method of brasquing, which differs somewhat from any of those described and recommended in the books, proved the most satisfactory. Pine charcoal is pulverized and screened through a 40-mesh sieve; the fine powder is then mixed with molasses and water in a suitable flat dish and kneaded with the fingers, until a homogeneous mass is obtained which, when squeezed in the hand, coheres into a lump and has not sufficient moisture to adhere to the hand.

Different proportions of molasses—the ordinary New Orleans molasses being used—and water were tried. Molasses alone with charcoal gave a sticky, shiny mass which cracked on the sides and bottom of the crucible, as it dried; two parts by volume of molasses and one of water behaved better, only the bottom cracking; three parts of molasses and one of water gave the desired mixture.

The crucible is moistened by dipping into water and quickly removing it. It is then partly filled with the charcoal paste and this is pressed with the fingers against the sides and the bottom, a slight excess of material being used; then a conical wooden plug, which has been dipped in water, is forced by hand into the charcoal

* *Traité des Essais*, 1847, vol. ii., p. 485.

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paste and turned round and round, the excess escaping over the edge of the crucible. A few light taps with a mallet help to make the lining more compact. The wooden plug has a mark, and when this reaches the upper rim of the crucible the plug is withdrawn with the same rotary motion with which it was forced in. It has the form of a long cone, the point being slightly flattened. Its size is such that when forced into the crucible down to the mark, the lining at the bottom will be $\frac{3}{4}$ inch thick. This thickness decreases until near the upper edge of the crucible, where it is only $\frac{1}{8}$ inch. It is then rounded off until even with the inner edge of the crucible, and the sides are polished with a strong test-tube to prevent any particles of tin from adhering, and to obstruct the filtering of fluxes through it. In drying, it is best to place the crucible upside-down. Otherwise the lining is liable to separate from the wall, especially if it dries too rapidly. When a little skill has been acquired, it takes four and a half hours, including the preparation of the mixture, to line fifty Battersea crucibles, half of size F and half of size C.

It will be seen that this lining is quickly made; it dries quickly without cracking and is hard when dry. After being in the fire, it is readily removed from the crucible, which can be used again, and is so compact and hard that it is difficult to break the thin upper edge with the fingers, while the thick lower part has to be broken with the hammer.

In summing up the subject of brasqued crucibles, it may be said that although charcoal linings are perfect from a chemical point of view, their use has practical objections for the separating of fine tin from fine slag in the pan. The results to be described will show that crucible-linings are in fact unnecessary.

The plumbago crucible stands between the naked and the brasqued crucibles, for while the action of the fluxes on the clay is weakened by the plumbago, the neutral character of the latter is slightly modified by the clay that binds it together. Plumbago crucibles are frequently used in the Cornish assay, but we shall see that this method is unreliable, and as the plumbago crucible is expensive, more need not be said about it.

Finally the porcelain crucible used by Levol has the same disadvantages as the naked clay crucible, although in a less marked degree.

II. Preparation of the Ore for the Experiments.

The more representative lode-tin not being available in sufficient quantities to be made the basis of the following experiments, stream-

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tin from the Nigger Hill district had to be resorted to. The cassiterite of the Bismarck district would have been the most desirable of all, as it is the purest. It is found in quartz and has comparatively no deleterious minerals occurring with it. This makes the separation of black tin from the gangue simple and perfect. The other lode-tins, as well as all the stream-tins of the Black Hills, have a number of minerals occurring with them which are difficult of removal, for instance, garnets.

The first pulverizing of the ore was performed with the distance of $\frac{1}{8}$ inch between the faces of the sample-grinder. The crushed ore was then screened through a 40-mesh sieve. At the second grinding the faces of the machine were brought somewhat nearer together and the sifting afterward repeated. The operation was continued until the ore had all become fine enough to pass through the sieve with the exception of some mica, which was thrown out. This gradual method was used in order to prevent the cassiterite from becoming too fine, as a large amount might thus have been lost in the washing of the ore. The pulp obtained was then roasted in a muffle to decompose any sulphides and arsenides that might be present. After roasting, and while still hot, it was thrown into cold water, to make silicates, not otherwise attacked by acids, decomposable. It was also hoped that the cassiterite might thus be made more readily reducible, an expectation which the event fulfilled. The ore was now boiled with nitro-hydrochloric acid to remove all soluble metallic compounds; this was continued until no more iron was perceived to go into solution. The ore was then washed with hot water and transferred to the gold pan, where the panning was carried on irrespective of the loss of some fine tin which floated off with the tailings, giving them a chocolate-brown color. As the main object was to obtain a pure black tin, the loss was unavoidable. The panning was continued until no more impurities were visible to the naked eye; under the microscope, however, particles of garnet with a little quartz were still visible. The black tin obtained was dried and intimately mixed by passing it repeatedly through a 20-mesh screen and rolling it on glazed paper. The now uniform ore was sampled down, and the final average sample pulverized in an agate mortar and put aside for chemical analysis. The weight of the purified black tin obtained, which was to form the basis of the experiments, was 7968 grammes. To find the size of the different particles, 500 grammes were taken and screened through different-

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sized sieves. The result of the mechanical analysis is given in the following table:

TABLE I.

Passed Through Sieve No.	Remained on Sieve No.	Grammes.	Per cent.
40	60	90	18
60	80	80	16
80	100	65	13
100		265	53
		500	100

The very brittle character of the cassiterite will be seen from this table, as 53 per cent. of the entire ore passed through a sieve of 100 meshes to the linear inch, although, as has been said, the greatest care was taken to grind in such a manner as to produce as small an amount of slimes as possible.

III. ANALYSIS OF THE PURIFIED BLACK TIN.

In making the analysis of the average sample of the purified black tin, the Rose method was followed, as modified by Chauvenet, who substitutes potassium carbonate for the sodium carbonate commonly used. Two separate determinations were made, taking 0.5 gramme in each case with 3 grammes of a mixture consisting of equal parts of potassium carbonate and sulphur, and fusing in a No. 1 Royal Berlin porcelain crucible. The first sample was heated three-quarters of an hour over a Berzelius alcohol lamp, and then for the same length of time over a gasolene lamp, after which the fused mass was treated with water in a beaker. Brown, gritty cassiterite now became visible at the bottom, showing that the decomposition was incomplete. The soluble potassium sulphostannate was then filtered off and the filtrate acidulated with sulphuric acid and put aside; the residue on the filter was treated with hot dilute nitric acid, and the iron, manganese and lime solution also kept. The residue was again fused with potassium carbonate and sulphur, but in a different way. The No 1 porcelain crucible was placed within a No. 3 porcelain crucible, and this within a No. 6 plumbago crucible, the bottom of which had been filled with refractory material,

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that the porcelain crucible might be nearly as high as the plumbago crucible. The three crucibles were then each covered with a lid, and the fusion performed in a pot-furnace, heated with anthracite coal at a moderate red heat. This lasted one hour. The fused mass was then treated as before, and the resulting decomposition found to be perfect. The fusion for the second analysis was made directly in the three crucibles, and was complete in one operation. To make sure of this, the fusion was repeated, and on acidulating the aqueous solution with sulphuric acid, the precipitate was found to have the milky-white color of sulphur.

The reason that the first fusion over the gasolene lamp was not successful, although prolonged until the fused mass became dry, must be sought in the fact that the air was imperfectly excluded. By fusing in a crucible which is inclosed in others, this exclusion can be entirely accomplished; in fact, upon removing the outer lid, when the crucible had been taken from the fire, but had not yet cooled, a strong odor of sulphurous acid was perceived, showing that not only had the air been excluded, but that the fusion had taken place in an atmosphere of sulphur.*

The analyses were both made by the usual methods for separating iron, manganese, and lime, and the results corresponded so closely that they can be expressed by a common average:

SnO_2	86.24 = 67.84 Sn
Fe_2O_3	2.62
Mn_2O_3	0.50
CaO	0.66
MgO	trace
Insoluble	9.39
	99.41

If this be compared with the analysis of the purest crystal of Nigger Hill stream-tin† that could be found:

SnO_2	93.06
Fe_2O_3	3.08
Insoluble	3.90
	100.04

it will be seen that the cleaning of the stream-tin, as described above,

* This use of the three crucibles was suggested by Dr. W. P. Headden, Professor of Chemistry at the Dakota School of Mines.

† Report of the Dakota School of Mines, 1888, p. 142.

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was very successful, the ore sample showing only 2.4 times as much insoluble residue as the crystal.

IV. The Assay in Detail.

The different methods of assaying black tin in the dry way may be grouped under two heads:

A. Those which aim at finding the actual amount of tin contained in the ore.

B. Those which determine approximately, how much metallic tin can be recovered by treatment on a large scale.

The first class divides, according to the character of the resulting metal, into methods (*a*) yielding a button of metallic tin, and those (*b*) yielding an alloy of tin with copper or iron.

Class A, a.

1. *The German Method of Assay.*—This method as given by Kerl* and by Balling† is as follows:

Five grammes of ore are intimately mixed with 0.75 to 1 gramme of charcoal dust and charged into a clay crucible; on top are placed 12.5 to 15 grammes black flux (or the substitute: 2 parts of potassium-sodium-carbonate, and 1 part of flour), with 1 to 1.25 grammes borax glass, then a salt cover, and finally a piece of charcoal. The crucible is covered, heated in a muffle or a pot-furnace at a moderate, gradually increasing, temperature, until the boiling has ceased, and then for from half to three-quarters of an hour at a white heat. The crucible is removed from the fire, broken, when cool, and the tin button weighed.

The process that takes place is simple. With the gradually rising temperature the tin becomes reduced to the metallic state by the charcoal with which it has been intimately mixed, while any ferric oxide contained in the cassiterite will be reduced only to ferrous oxide and taken up by the slag. At a certain stage the black flux, or the flour of the substitute, becomes decomposed, the result being that finely divided carbon is uniformly distributed through the flux. This hinders any particles of stannic oxide from combining with the alkali when fusion begins, and assists the reduction of particles of ore that have not been been completely converted into metal by the charcoal. The active fluxes—potash, soda, borax,—combine with the

* *Metallurgische Probirkunst*, Leipsic, 1882, p. 412.

† *Die Probirkunde*, Brunswick, 1879, p. 391.

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gangue contained in the ore and form a slag; the neutral salt cover assists in making the slag liquid and thus favors the collecting of tin particles, and prevents prills from adhering to the sides of the crucible; the charcoal, finally, being slowly consumed, furnishes the reducing atmosphere above the charge. When the reactions have taken place and the fusion has become tranquil, the different parts settle according to their specific gravities: at the bottom, the tin button; next, the borax slag; above this the salt slag; any unconsumed charcoal remaining on top.

In carrying out the assays, the above method was closely followed. The crucibles were size F, of the Battersea make; the charcoal was made to pass a 40-mesh sieve before it was mixed with the ore; the black flux substitute—2 parts of potassium carbonate and 1 part of flour, to which 1 gramme of borax glass had been added,—was used for flux.

The charge melted readily and the slag was very thin, showing, when cool, a smooth surface. The crucible was not corroded by the fluxes, and no particles of tin were visible at the sides. On breaking it the salt slag separated readily from the borax slag, the former being coarsely crystalline and brittle. It had a resinous luster, was subtranslucent, and of a dark-purple color. This color must be attributed to finely divided carbon resulting from the charcoal on top. On dissolving the slag in water, a fine black slime remained. This proved to be pure carbon, and the solution showed only a trace of iron. The borax slag was vitreous, hard, subtranslucent, and olive-green. The button separated well from the slag, was white, bright, easily cut, and malleable; scrapings were not attracted by the magnet, showing the absence* of iron. The tin, however, did not all collect in the button, as usually happens with tin assays. Prills remained suspended in the slag and some adhered to that part of the crucible in contact with it. In the assays the salt slag, the borax slag, and the lower part of the crucible were ground together and screened through a 40-mesh sieve. The scales of each set of assays were weighed together and the fine pulp of the same set panned and the resulting fine tin weighed. It seemed advisable to do it in this manner, because the amount of scales and siftings in the separate assays was so small that it might easily have caused an error in the results.

If the percentage of tin recovered in the assays Nos. 1, 2, 3,

* Berthier, *Traité des Essais*, 1847, vol. ii., p. 473.

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67.58 per cent., and in Nos. 4, 5, 6, 67.46 per cent., be compared with the actual amount of tin present, 67.84 per cent., it will be seen

TABLE II.

No. of Assay.	Resulting Tin.							
	Total.		In Button.		In Scales.		In Siftings.	
	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.
1	3.379	67.58	3.37	67.40	.....	.....	.....	.....
2	3.379	67.58	3.37	67.40	.....	.....	.....	.....
3	3.379	67.58	3.37	67.40	.....	.....	.....	.....
Average	3.379	67.58	3.37	67.40	0.001	0.02	0.008	0.16
4	3.380	67.60	3.31	66.20	.....	.....	.....	.....
5	3.370	67.40	3.30	66.00	.....	.....	.....	.....
6	3.370	67.40	3.30	66.00	.....	.....	.....	.....
Average	3.373	67.46	3.303	66.06	0.015	0.30	0.055	1.10

that the results obtained by this method are all that could be desired. The two averages differ only 0.12 per cent., which is also very favorable. That assays Nos. 1, 2, 3 agree accurately is mere accident. It will be seen that 67.40 per cent. collected in the button, and that only 0.28 per cent. was scattered through the slags and adhered to the crucible. Assays Nos. 4, 5, 6 agree within 0.2 per cent. Less tin was collected in the button than in Nos. 1, 2, 3, and the total result is slightly lower, although still very good, *i.e.*, accurate within 0.5 per cent.

To see what influence would be produced by variations of time and temperature, assays Nos. 7 to 10 were made, increasing the time to two hours with Nos. 7, 8, 9, 10, and keeping Nos. 9 and 10 hotter than Nos. 7 and 8, and these last hotter than Nos. 1 to 6.

Table III. shows that the loss in tin increases if the crucibles are left too long in a very hot fire; the buttons, however, remaining free from iron.

Although the results in the naked crucible were entirely satis-

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factory with normal conditions of temperature and time, the chalk-lined crucible was now used, to see what effect would be produced

TABLE III.

No. of Assay.	Resulting Tin.							
	Total.		In Button.		In Scales.		In Siftings.	
	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.
7	3.320	66.40	3.25	65.00	.....	.....	.....	.....
8	3.310	66.20	3.24	64.80	.....	.....	.....	.....
Aver...	3.315	66.30	3.245	64.90	0.015	0.30	0.055	1.10
9	3.000	60.00	2.93	58.60	.....	.....	.....	.....
10	2.850	57.00	2.78	55.60	.....	.....	.....	.....
Aver...	2.925	58.50	2.855	57.10	0.015	0.30	0.055	1.10

upon the assay. Assays Nos. 11, 12, 13 were made in the same way as Nos. 1 to 6, keeping the temperature reasonably low. No difference could be perceived in examining the crucible, slags, and button, except that the upper part of the crucible was more tarnished than with the naked one.

TABLE IV.

No. of Assay.	Resulting Tin.							
	Total.		In Button.		In Scales.		In Siftings.	
	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.
11	3.393	67.86	3.38	67.60	.....	.....	.....	.....
12	3.373	67.46	3.36	67.20	.... .	.....	.....	.....
13	3.363	67.26	3.35	67.00	.....	.....	.....	.....
Average	3.376	67.53	3.363	67.27	0.001	0.02	0.012	0.24

Table IV. shows an average result which does not differ much

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from that obtained in the naked crucible, but the assays vary as much as 0.6 per cent.

If time and temperature are increased, the difference from the results with the naked crucible becomes much greater. See Table V.

TABLE V.

No. of Assay.	Resulting Tin.							
	Total.		In Button.		In Scales.		In Siftings.	
	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.
14	3.456	69.12	3.450	69.00	.....	.....	.....	.....
15	3.456	69.12	3.450	69.00	.....	.....	.....	.....
16	3.446	68.92	3.440	68.80	.....	.....	.....	.....
17	3.446	68.92	3.440	68.80	.....	.....	Average. 0.006	Average. 0.12
18	3.441	68.82	3.435	68.70	.....	.....	.....	.....
19	3.421	68.42	3.415	68.30	.....	.....	.....	.....
20	3.391	67.82	3.385	67.70	.....	.....	.....	.....
21	3.371	67.42	3.365	67.30	.....	.....	.....	.....
22	3.291	65.82	3.285	65.70	.....	.....	.....	.....

With the exception of No. 22, all the buttons contained iron, some of the results being even then too low (Nos. 20 and 21), and others too high (Nos. 14 to 19). Apparently the lime, by entering the slag, assisted in carrying the iron into the tin. That, nevertheless, some tin was slagged is to be accounted for by the fact that there was not enough iron to prevent it, for metallic iron,* although, if present in sufficient quantity, it will precipitate the tin completely, if in too small amount, will only reduce the stannic oxide to stannous oxide, which of course enters the slag.

Kerl† mentions the fact that by first heating with charcoal alone in the crucible and then adding the fluxes, the scorification of tin will be lessened. The reason of this is apparent. If the stannic oxide be reduced to the metallic state, the fluxes then added will, upon fusion, simply assist to collect the fine particles of metallic tin into

* Berthier, *Traité des Essais*, vol. ii., p. 459.

† *Metallurgische Probirkunst*, 1879, p. 483.

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a button, and the danger that stannic or stannous oxide may be carried off in the slag, is excluded. But other more powerful causes counteract the good results which might be expected from this modification of the German assay. The most important is the presence of iron in the metal. Although tin is reduced sooner than iron to the metallic state, there is no direct indication by which the assayer can know when to stop with the reduction and when to add the fluxes. If the reduction is carried too far, the iron also will be reduced and will enter the button; if not carried far enough, so that tin still remains unreduced, the main advantage of the modified method is lost. Another drawback is that the fluxes are added to the hot reduced ore in the crucible. This creates a liability to loss of ore by dusting.

In carrying out the experiments, the proportions used were the same as with the regular method. The crucibles were placed in the hot fire, and counting from the time that the ore appeared dark red, it was exposed for a quarter of an hour to a strong heat; then the fluxes were added in their regular order and the charge heated for one hour.

The behavior in the crucible and the general appearance of crucible and slags were not different from that with the regular method. The color of the salt slag, however, was light-green instead of dark-purple, that of the borax slag dark olive-green instead of the usual lighter shade. The tin buttons did not separate well from the slag, as their surface was very uneven. In some places were slight depressions or even cavities; in others, the button was covered with small beads of metal adhering partly to it and partly to the borax slag. Some of the buttons were bright and more or less malleable and free from iron. Others, however, contained so much iron that they had not the usual white color and bright appearance, and cracked when hammered flat to the size of a dime.

Two double sets of experiments were carried out, the one in a naked crucible at a normal and then at a high temperature, the other in a chalk-lined crucible under the same conditions. The results obtained in the naked crucible are given in Table VI.

The average result of assays Nos. 23, 24, 25, carried out at a normal heat, is over 1 per cent. lower than that obtained by the regular method and there is a difference of 2.6 per cent. between the highest and the lowest of the results. The buttons, however, were all free from iron. Those from assays Nos. 26, 27, 28, carried out at a high temperature, show the presence of iron in quite a marked

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degree. There was no regularity whatever: No. 26 took up so much iron that the button weighed more than the tin obtained by chemical

TABLE VI.

No. of Assay.	Resulting Tin.							
	Total.		In Button.		In Scales.		In Siftings.	
	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.
23	3.363	67.26	3.33	66.60	.....	.....	.....	.....
24	3.333	66.66	3.30	66.00	.....	.....	.....	.....
25	3.253	65.06	3.22	64.40	.....	.....	.....	.....
Average	3.316	66.32	3.283	65.66	0.002	0.04	0.031	0.62
26	3.484	69.68	3.460	69.20	.....	.....	.....	.....
27	3.334	66.68	3.310	66.20	.....	.....	.....	.....
28	3.259	65.18	3.235	64.70	.....	.....	.....	.....
Average	3.359	67.18	3.335	66.70	.....	.....	0.024	0.48

analysis; Nos. 27 and 28 also took up iron, and at the same time gave some tin to the slag.

The results obtained by this modified method are therefore unsatisfactory. They become still more so, if the assay is carried on in the chalk-lined crucible. This can be seen from Table VII.

Of Nos. 29, 30, 31, made at a low temperature, one assay—No. 29—contained iron and between the other two there is a difference of 7.4 per cent. This discrepancy was probably caused by the large amount of tin carried off in the slag of No. 31. Under the caption "siftings" there is a blank. As siftings, there remained in the pan a heavy brown-black slag which, when examined in the wet way, proved to be rich in tin. The assays Nos. 32, 33, 34, carried on at a high temperature, gave buttons which were all strongly ferruginous; they also showed great irregularity. Judging from the results obtained by the modified method—Nos. 23 to 32—the preference must be given to the regular German method, although, as above stated, it would seem rational to suppose that the loss in tin would be

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smaller, if the black tin were reduced to the metallic state, before the flux is added and the fusion begins.

Mitchell* gives a method, which may be mentioned in connection

TABLE VII.

No. of Assay.	Resulting Tin.							
	Total.		In Button.		In Scales.		In Siftings.	
	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.
29	3.419	68.38	3.41	68.20	.....	.....	.....	.....
30	3.339	66.78	3.33	66.60	.....	.....	.....	.....
31	2.969	59.38	2.96	59.20	.....	.....	.....	.....
Average	3.242	64.84	3.233	64.66	0.009	0.18	.....	.....
32	3.425	68.50	3.390	67.80	.....	.....	.....	.....
33	3.365	67.30	3.330	66.60	.....	.....	.....	.....
34	3.205	64.10	3.170	63.40	.....	.....	.....	.....
Average	3.332	66.63	3.297	65.93	0.001	0.02	0.34	0.68

with the German assay. He mixes 400 grains ore, 100 argol, 300 sodium carbonate, 50 lime, and charges in a crucible so large that it need only be half filled, then gives a cover of sodium carbonate and 200 grains of borax; heats gently and keeps for at least twenty minutes at a dull red heat, increasing the temperature, until tranquil fusion takes place.

The experiments were carried out as directed. A beautiful bottle-green slag resulted. It was as glassy as obsidian and covered with a thin slag of a lighter green color. The results from three consecutive assays were so discouraging that no further attempts were made in that direction. Nearly all the tin had been taken up by the slag, the only apparent reason being the intimate mixing of ore and flux.

2. *Fusion with Calcium Fluoride.*—There are three methods described under this head, in all of which fluorspar is used as the only or principal flux. They are recommended for ores containing silica only.

* *Manual of Assaying*, New York, 1881, p. 481.

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Fluorspar is not readily fusible; in fact, it cannot be melted at temperatures produced in a muffle-furnace. Assays requiring it as a flux must therefore be carried on in a pot-furnace. While difficult of fusion, it is liquid when melted, and assists to liquefy compounds which have a refractory character, by holding in suspension the particles which cannot be completely melted. It is therefore very satisfactory, if a high temperature is required. Berthier* was the first to suggest that a loss in weight takes place if calcium fluoride and silica are heated together. In later publications† the formula $2\,CaF_2 + 2SiO_2 = SiF_4 + Ca_2SiO_4$ is found with the statement that part of the silica escaped as silicon fluoride. Percy‡ has shown by experiments that while a loss in weight does take place, the formula is wrong, as it expresses more than three times the actual amount lost. It would seem, therefore, that the basis on which these methods are founded is not a correct one. Owing to the high temperature, there is always danger of losing tin through volatilization, although§ in a charcoal-lined crucible this may be prevented by entirely excluding the air. In regard to scorification, it seems doubtful whether it can be avoided so as to give satisfactory results.

(1.) Ricketts|| recommends mixing 10 grammes of ore with 10 to 20 grammes of fluorspar or cryolite. He then says, "charge in a charcoal-lined crucible, which is first covered with charcoal and then luted with clay; heat strongly for about one hour; remove carefully from the fire and tap gently. Treat the button as an alloy afterwards." These instructions were closely followed, fluorspar being used and the time of exposure to heat lengthened, as previous experiments in charcoal-lined crucibles had shown that considerable time is required for the heat to penetrate the lining.

Three experiments were made, resulting in a gray, slightly fritted mass, which contained white tin and black tin mixed with fluorspar. This gave, when crushed in a porcelain mortar and treated with water, gray slimes, and, as residue, the mixture of white and black tin. The complete reduction of the stannic oxide was probably prevented by the intimate admixture of fluorspar. It could scarcely have been caused by insufficient heat. A hot anthracite fire was used; the furnace was 12 x 12 inches, the crucible resting on 3 inches of

* *Traité des Essais*, 1847, vol. i., p. 436.

† *Eg*, Balling, *Metallurgische Chemie*, Bonn, 1832, p. 88.

‡ *Metallurgy of Fuel*, etc., London, 1875, p. 80.

§ Berthier, *Traité des Essais*, 1847, vol. ii., p. 458.

|| *Notes on Assaying*, 1886, p. 88.

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coal and being covered by 3 inches of coal (to the lower edge of the flue). The draught was strong, the total time of exposure to heat 2 hours.

(2.) Ricketts* gives a similar charge for a chalk-lined crucible: ore 10 grammes, fluorspar 10 grammes, powdered charcoal 2 grammes, a salt and charcoal cover, and recommends a hot fire.

This method produced negative results. The charge at the bottom and sides of the crucible was fused; it was gray, showing a crystalline structure; at the bottom the fused rim was $\frac{1}{8}$ inch thick, decreasing to $\frac{1}{16}$ inch higher up; inside this shell was a fritted mass of a color from dark-gray to black, consisting apparently of charcoal and fluorspar, with fine particles of tin dispersed through it, and covered by a pumice-like gray slag, probably a mixture of salt and fluorspar.

(3.) Balling† recommends mixing 5 grammes of ore with 2.5 grammes of fluorspar and 2.5 grammes of lime, charging the mixture in a charcoal-lined crucible, and treating as in the old German method of assaying iron-ores in the dry way; that is, heating for an hour and a half, the last fifteen or twenty minutes at a white heat.

The result of several fusions was a slightly sintered mass of a grayish-white color collected into a button. Through it were distributed prills of metallic tin of different sizes, the largest being about the size of a small pin-head. Another negative result.

No pains were spared to make these experiments with fluorspar as thorough as possible, and the failure to obtain satisfactory results must have been due to one of two things. Either the ore used was not suited to that mode of treatment, or important particulars were omitted from the directions given.

3. *Fusion with Potassium Cyanide.*—This very valuable method of assaying tin-ores is given by Mitchell.‡ In describing his manner of operating, he says he uses a 3-ounce Hessian crucible, rams into its bottom a $\frac{1}{2}$-inch layer of potassium cyanide, adds the charge, consisting of 100 grains of ore mixed with from four to five times its weight of powdered potassium cyanide, and gives a cover of the same flux. He then heats at a moderate temperature for ten minutes, removes the crucible, taps it gently, and allows it to cool. He dissolves the slag in water to see if any reduced metal or heavy

* *Notes on Assaying*, 1886, p. 89.

† *Die Probirkunde*, 1879, p. 391.

‡ *Manual of Assaying*, 1881, pp. 481 to 483.

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particles of the original ore are present, and says that, with ordinary care, the results are accurate to ½ per cent.

The basis on which the process rests can be expressed by the formula: $SnO_2 + 2KCy = Sn + 2KOCy$. Potassium cyanide, being more powerful than any other reagent, acts more quickly and also at a lower temperature, thus reducing the volatilization of tin to a minimum. The disadvantages are its cost and its very poisonous character. The extra expense need hardly be taken into consideration, as it is very nearly counterbalanced by the saving in time, the difference being fully made up by the superiority of the results obtained. Its poisonous character requires that special care should be taken in handling it. The mortar ought to be covered with a wooden lid, having a hole through which the handle of the pestle passes, and a cloth placed over this to avoid dusting. A sponge or handkerchief should protect nose and mouth while pulverizing, weighing, and mixing the reagent with the ore. The crucible should be placed to cool under a good draught, and care taken to finish the breaking of crucibles and handling of cyanide slags before the fingers have occasion to become moist. If these precautions are observed there is no danger whatever, as the writer can testify from repeated experience.

In the experiments the ordinary chemically pure potassium cyanide —98 per cent. KCy and 2 per cent. KOCy—was first used; then the effects produced by the different impurities that are apt to be found in commercial cyanides were studied; and, lastly, the commercial cyanides themselves were tested. The regular method given by Mitchell was varied in different ways to see what effect modifications from other methods would have on the reagent.

In regard to temperature and time required for the assay, Mitchell says, "a moderate fire," which will keep the charge at a "steady fusion" for "ten minutes." It was found that a charge can be kept at a steady continuous fusion for twenty minutes, and the temperature be too low to complete the chemical reaction. Small globules of tin can be seen floating about in the slag, and these will not be collected into one button when the crucible is broken. On the other hand, the temperature may be raised so high that the charge will boil, and the fusion still be steady and continuous. The tin will be reduced, but will not collect into one button, and the assay will again be imperfect. The best way to regulate the temperature would seem to be the following: Begin with a hot fire and keep it at the highest point to which potassium cyanide can be heated without beginning to boil and evolve heavy fumes. This temperature is easily recogniz-

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able after a few assays. Ten minutes is a very good average for the time, beginning when the charge is liquid and brown-red. An assay may be considered finished when the pure upper slag has become so transparent that the impurities contained at the bottom of the crucible are visible through it. The tin will then have collected into one button beneath the lower slag, and very few, if any, prills will be found.

The experiments with potassium cyanide were as follows (1 to 11):

(1.) 5 grammes of ore, mixed with 20 grammes of potassium cyanide, were charged in a Battersea crucible, size F, into the bottom of which had been rammed 5 grammes of cyanide; the charge was covered with 5 grammes of cyanide; time given, ten minutes.

The slag was as liquid as water. When cold its surface was smooth, and had the shining luster of porcelain. The crucible was not corroded, and the upper part was free from tin. On breaking the crucible it was found that the potassium cyanide had filtered through nearly the entire bottom and had blackened it, the blackness diminishing on the sides and disappearing near the top. The fracture of the upper slag was coarsely granular; it was opaque, milk-white, brittle, and soluble in water. On the bottom of the crucible was an amorphous, uneven slag; it had a resinous luster, was subtranslucent, of light-green color, harder than the white slag, and insoluble in water. It was charged with the impurities contained in the ore. The button separated well from the slag, was white, bright, easily cut, malleable, and free from iron.

To examine the slags and lower part of the crucible for prills of tin, they were placed in hot water. The white slag dissolved readily, and the green, as a rule, separated easily from the crucible. The solution of potassium cyanide and cyanate was removed by decanting, and the pieces of crucible taken out and dried apart from the residue in the dish. Both were then crushed separately and passed through a 60-mesh sieve. The resulting fine pulp was panned, and the siftings weighed with their scales, the slags and crucibles of one series of assays being worked up together, as the weights of the resulting buttons were close enough to justify the proceeding.

Table VIII. shows that the average result is 0.54 per cent. lower than that found by chemical analysis, and the percentage of tin in slags and crucible—1.10 per cent.—rather higher than would have been expected. The greatest discrepancy, however, in the total weights is only 0.03 gramme, but expressed in percentage it amounts to 0.6

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per cent. It seemed, therefore, more advisable to use 10 grammes of ore, as in the next experiments.

TABLE VIII.

No. of Assay.	Resulting Tin.							
	Total.		In Button.		In Slags.		In Crucible.	
	Grms.	Per cent.	Grms.	Per cent.	Grms.	P'r c'nt.	Grms.	P'r c'nt.
35	3.385	67.70	3.330	66.60	.....	.....	.....	.....
36	3.365	67.30	3.310	66.20	.....	.....	.....	.....
37	3.355	67.10	3.300	66.00	.....	.....	.....	.....
38	3.355	67.10	3.300	66.00	.....	.....	.....	.....
Aver...	3.365	67.30	3.310	66.20	0.005	0.10	0.050	1.00

(2.) The charge consisted of ore, 10 grammes; potassium cyanide, 40 grammes; 5 grammes cyanide for bottom of crucible and 5 grammes cyanide for cover; crucible, size F; time, fifteen minutes. The only difference between the first and second assays is that in

TABLE IX.

No. of Assay.	Resulting Tin.							
	Total.		In Button.		In Slags.		In Crucible.	
	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.
39	6.752	67.52	6.685	66.85	.....	.....	.....	.....
40	6.752	67.52	6.685	66.85	.....	.....	.....	.....
41	6.752	67.52	6.685	66.85	.....	.....	.....	.....
42	6.742	67.42	6.675	66.75	.....	.....	.....	.....
Average	6.749	67.49	6.682	66.82	0.060	0.60	0.007	0.07

the latter the button was covered with a larger quantity of green slag showing (as the crucibles were of the same size as before) a thicker layer. In assays Nos. 35 to 38 this heavy green slag had collected around the button, filling the space between it and the

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crucible, and not quite reaching the top of the button which was covered by the pure white slag. It would therefore seem advisable to use, for a 10-gramme assay, a crucible having a larger base than the size F of the Battersea make. The button could then lie on the bottom and be surrounded by the impure slag, the top being covered by pure white slag, which helps reduction and the successful collecting of the metal in one button. If the assays are made in the muffle, the 10- and 20-gramme, or Colorado A and B crucibles, used for lead and slag assays, might be very suitable for the purpose.

The total average differs here only 0.35 per cent. from the chemical analysis; three assays agree perfectly, and the fourth shows only 0.01 gramme or 0.1 per cent. difference, which is very close. The percentage of tin in slags and crucible is half as much as that in the previous assays (Table VIII.), suggesting that the first assays, Nos. 35 to 38, might have been benefited by a longer exposure to the heat.

(3.) In assaying ores in the dry way it has been, and still is, very common to give a salt cover. To try what effect this would have, the two foregoing sets of experiments were repeated, adding the salt.

It was found that the charges melted less rapidly and were not so thin as before; when cold, the surface of the slag was uneven and rough; the crucible was corroded, but less soaked with potassium cyanide, the upper part showing no prills of tin; the slag showed a very coarsely granular structure; it was opaque, milk-white, but somewhat dull in comparison with that of the pure cyanide; no other differences were observable.

The low results shown by the averages were startling. They could not be attributed to any action of the sodium chloride nor to the small amount of silicate derived from the corrosion of the crucible. The salt was examined and found to be rich in sulphuric acid; this explained the loss in tin (see p. 32). Fine Michigan table-salt was then tested, and sulphuric acid found to be present here also. Therefore, nothing further was attempted with salt. No special advantage is apparent in its use with potassium cyanide, as the slags are thinner without it and form a better "wash" for the sides of the crucible than the less readily fusible salt. If it is to be used at all, recourse must be had to Ohio salt, which, according to the *Report of the Geological Survey of Ohio*, "Economic Geology," 1888, vol. vi., page 663, is free from sulphates, while Michigan salt always contains them.

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(4.) Ricketts recommends the chalk-lined crucible for the assay with potassium cyanide. He gives the charge, 10 grammes of ore and 40 grammes of potassium cyanide, with half of the flux in the bottom of the crucible and the other half mixed with the ore, then the usual

TABLE X.

No. of Assay.	Resulting Tin.							
	Total.		In Button.		In Slags.		In Crucible.	
	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.	Grms.	Per cent.
43	3.266	65.32	3.200	64.00	.....	.....	.....	.....
44	3.231	64.62	3.165	63.30	.....	.....	.....	.....
45	3.226	64.52	3.160	63.20	.....	.....	.....	.....
46	3.176	63.52	3.110	62.20	.....	.....	.....	.....
Average	3.225	64.49	3.159	63.17	0.020	0.40	0.046	0.92
47	6.505	65.05	6.430	64.30	.....	.....	.....	.....
48	6.405	64.05	6.330	63.30	.....	.....	.....	.....
49	6.235	62.35	6.160	61.60	.....	.....	.....	.....
50	6.220	62.20	6.145	61.45	.....	.....	.....	.....
Average	6.341	63.41	6.266	62.66	0.065	0.65	0.010	0.10

cover of cyanide, to which is added a salt-cover, and heats for fifteen minutes. These directions were followed, only omitting the salt cover.

It was expected that the chalk-lining would at least partly prevent the cyanide from filtering into the crucible, but no difference could be observed, the crucible being just as black as if it had had no chalk-lining.

The structure of the slag was slightly coarser; it was more uneven in fracture and of a bluish-gray color, instead of the usual pure milk-white.

The average of the results is 2.1 per cent. lower than that obtained by the chemical analysis, and 1.75 per cent. lower than the average of the corresponding 10-gramme assays, Nos. 39 to 42. The

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separate assays also show a difference of 1.9 per cent., which is very unfavorable. The first reason for this that suggested itself was the possible presence of gypsum in the chalk, but no sulphates were found. There remains, therefore, only the presence of lime to account for the slagging of tin.

By a mistake in assay No. 54, the entire 40 grammes of potassium cyanide were mixed with the 10 grammes of ore, and charged with it. No button whatever was obtained. The bottom of the crucible was

TABLE XI.

No. of Assay.	Resulting Tin.							
	Total.		In Button.		In Slags.		In Crucible.	
	Grms.	Per cent.	Grms.	Per cent.	Grms.	P'r c'nt.	Grms.	P'r c'nt.
51	6.648	66.48	6.625	66.25	.....	.....	0.017	0.17
52	6.618	66.18	6.595	65.95	.....	.....	0.017	0.17
53	6.458	64.58	6.435	64.35	.....	.....	0.017	0.17
54	.....	.....	.....	.....	.....	.....	.....	.....
Ave. ...	6.574	65.74	6.551	65 51	0.006	0.06	0.017	0.17

covered with a spongy, greenish-gray mass, and this by the ordinary white slag of potassium cyanide. This shows the necessity of having a good layer of potassium cyanide on the bottom of the crucible, probably on account of the tendency the tin has not to collect in a single button. If the crucible is filled with a uniform mixture of ore and cyanide a number of small buttons will be produced, mixed with the slag. If a layer of pure potassium cyanide cover the bottom of the crucible, the particles of tin will sink into it and form on the undisturbed bottom a nucleus, around which other particles can collect and thus take up gradually the most of the tin. The effect is similar if charcoal be mixed with the potassium cyanide (see 5 to 7).

(5.) The regular method of assay with potassium cyanide was modified in the same way as the German method, *i.e.*, the stannic oxide was first reduced by charcoal, and the potassium cyanide then added to collect the particles of metallic tin into one button. It was expected that the iron would be taken up by the potassium cyanide instead of alloying with the tin. The result was tested by dissolv-

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ing the slag in water, filtering the solution, acidulating the filtrate with hydrochloric acid, and boiling to expel the hydrocyanic acid formed. It was then tested with a mixture of ferrous and ferric salt which gave the reactions for potassium ferro- and ferri-cyanide; another part of the filtrate was evaporated to dryness, treated with nitro-hydrochloric acid, again evaporated to expel all cyanogen, and taken up with hydrochloric acid; on adding ammonium hydroxide, the reddish-brown ferric hydroxide was precipitated, showing again the presence of iron.

In the experiments, 5 grammes of ore were mixed with 1 gramme of charcoal, placed in a naked crucible, size F, and heated for a quarter of an hour after the charge had become dark-red; then 20 grammes of potassium cyanide were added; after this thirteen minutes were required before quiet fusion was obtained.

The slags had the usual characteristics of the pure cyanide, but black patches of carbon were irregularly dispersed through them. The buttons were white, very bright, malleable and free from iron; they were, however, rough and covered with bead-like excrescences of tin.

TABLE XII.

No. of Assay.	Resulting Tin.							
	Total.		In Button.		In Slags.		In Crucible.	
	Grms.	Per cent.	Grms.	Per cent.	Grms.	P'r c'nt.	Grms.	P'r c'nt.
55	3.223	64.46	3.100	62.00	.....	.....	.....	.....
56	3.202	64.04	1.310	26.20	1.892	37.84	See slags.	See slgs.
57	3.198	63.96	3.075	61.50	.....	.....	.....	.....
58	3.198	63.96	3.075	61.50	.....	.....	.....	.....
Aver...	3.205	64.11	.....	.....	0.033*	0.66*	0.090*	1.80*

These results are very unsatisfactory. The average gives 3.73 per cent. less tin than is present. It is 3.20 per cent. lower than that of the corresponding 5-gramme assays, Nos. 35 to 38, and 2.21 per cent. lower than that of the modified German method, Nos. 23 to 25. On comparing the separate weights of the tin-buttons obtained, it

* Except No. 56.

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will be seen that the tendency of the tin not to collect in one button is enhanced by the presence of the charcoal with which the ore had been mixed. The only result of positive interest obtained by the experiments is the fact that the iron of the black tin, reduced to the metallic state, goes into the slag and not into the button, as in the modified German assay.

Similar experiments carried on in a charcoal-lined crucible were equally unsatisfactory.

(6.) The same method, only using a porcelain crucible, bears the name of Levol's method. Kerl* and Balling† give it as follows: 2 grammes of ore are mixed with 20 per cent. of charcoal in a porcelain crucible and heated for from fifteen to thirty minutes in a muffle; 0.50 to 0.75 gramme of potassium cyanide are added, and the heating continued another five minutes. They say that the results are accurate to from 0.3 to 0.5 per cent.

It was found that it is necessary to grind ore and charcoal very fine, and that an excess of charcoal must be avoided.

Two grammes of ore were mixed with 0.5 gramme of charcoal and heated for half an hour in the muffle at a good scorification temperature. The crucibles were then removed, allowed to cool, 1 gramme of potassium cyanide added, and the crucibles returned to the muffle for ten minutes.

They were all blackened and somewhat corroded; the slag was of a light-gray color from the charcoal, and there was a small amount of green slag; in none of the assays had the tin collected in one button; it varied in size from a button down to a fine gray powder. The crucibles were now placed in hot water, the small buttons picked out, and the fine tin separated from the charcoal and insoluble green slag by washing. This is the uncertain part of the assay. The fine tin separates with difficulty from the green slag. If this is entirely removed, some tin will be lost, and the resulting weight will be too low; if all the tin is recovered, some slag remains with it, and the result is too high. Thus, the assayer must choose between two evils, and this causes a feeling of uncertainty. Any slight mistake is multiplied by fifty when the result is expressed in percentage. This is illustrated in Table XIII.

The average is too low, showing loss from excessive care in washing to obtain pure tin. The greatest discrepancy between the single

* *Metallurgische Probirkunst*, 1882, p. 483.

† *Die Probirkunde*, 1879, p. 392.

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results is 2.25 per cent. This shows how difficult it is to effect a satisfactory separation of fine insoluble slag and fine tin. During the past two years the writer has made a number of assays according to this method. The results obtained from duplicate assays only agreed well when all the tin had collected in one or more large

TABLE XIII.

No. of Assay.	Resulting Tin.	
	Grms.	Per cent.
59	1.350	67.50
60	1.335	66.75
61	1.305	65.25
Average	1.330	66.50

buttons; they were never satisfactory if the tin was present in the form of fine powder. It was not discovered why, in one case a button, in another a powder, should be obtained. The statement made by Kerl and Balling that the results of this method are accurate within 0.5 per cent. needs modification, as it certainly does not apply to all tin-ores. Probably those of the Black Hills contain impurities that were not present in the ores Kerl and Balling have reference to.

(7.) The last experiments made with potassium cyanide followed T. W. E. David in his *Report on the Geology of the Vegetable Creek Tin-Mining Field*, Sydney, New South Wales, 1887, p. 153. Mr. David says that at the Glen Smelting Works, at Tent Hill, the potassium cyanide is mixed with charcoal. The only reason that can be brought forward for this modification is, that the potassium cyanide used must have been very impure. Thus, in order to increase its reducing power, charcoal was added.

Five grammes of ore were fused with a mixture of 30 grammes of potassium cyanide and 1 gramme of charcoal; only two-thirds of the flux were mixed with the ore, one-third being rammed into the bottom of the naked crucible.

Contrary to expectation, the charge fused as readily as if it contained no charcoal, and came very quickly to a quiet fusion. On breaking the crucible, no button was to be found, but only fine prills

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of tin disseminated through a pumice-like, dark-gray slag, containing irregular streaks colored black by carbon. This shows again the necessity, alluded to above, of having in the bottom of the crucible a layer of pure potassium cyanide for the reduced particles of tin to collect in and form a button.

In the preceding experiments, the ordinary c. p. potassium cyanide was used, as it seemed important that the results obtained should be based on a reagent whose composition was definitely known. But this pure cyanide is somewhat expensive ($1.90 a pound), and to see whether its use was absolutely necessary, the different commercial grades were tested. First, however, c. p. cyanide was mixed with cheaper fluxes, such as pure potassium and sodium carbonate, to obtain, if possible, a flux which would have the same action as the pure cyanide, but be considerably cheaper. The effect produced by potassium sulphate was also investigated.

(8.) The effect of potassium carbonate on the cyanide assay was tested by fusing in a naked crucible 5 grammes of ore with 30 grammes of potassium cyanide, to which had been added different quantities of this flux. One-third of the flux-mixture was rammed into the bottom of the crucible, and the rest mixed with the ore, except a small quantity kept to be used as a cover. The average time required to complete the assay was ten minutes from the time the charge began to fuse.

The charges behaved during the fusion just as if pure cyanide had been used. The crucibles remained uncorroded. On breaking, it was found that with the increase of potash in the mixture, the crucible was less soaked with cyanide; in the last assay, with 41.17 per cent. potassium carbonate, the bottom was blackened only to the depth of $\frac{1}{8}$ inch. The top of the slag was, in all the assays, as smooth as with pure cyanide. The color had, however, changed. With small quantities of potash, the slag, on the surface as well as on the fracture, had a bluish tinge which grew deeper until, in the last assay, it became a decided blue-gray color. The structure of the slag became more and more granular, until it was so coarse as to have the appearance of granulated sugar cemented together. Finally, with the increase of potash, the luster of the slag decreased, while at the same time it grew harder and tougher. The buttons of tin obtained did not differ in appearance from those with pure cyanide.

In Table XIV. two points are striking: First, that the amount of tin obtained grows smaller with the increase of potassium carbonate,

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and, secondly, that no tin is recovered in the slags or in the crucible. Both facts are probably due to the same cause,* viz., that metallic tin decomposes alkali carbonates, forming stannates which are not readily fusible. In fact, Berthier† says: "By fusing one part of

TABLE XIV.

No. of Assay.	Charge.				Resulting Tin.	
	Ore.	KCy.	K_2Co_3.			
	Grms.	Grms.	Grms.	Per cent. in Mixture.	Grms.	Per cent.
62	5	30	3	9.09	3.325	66.50
63	5	30	6	16.66	3.300	66.00
64	5	30	9	23.07	3.275	65.50
65	5	30	12	28.57	3.263	65.26
66	5	30	15	33.33	3.235	64.70
67	5	30	18	37.50	3.230	64.60
68	5	30	21	41.17	3.190	63.80

metallic tin with four or five parts of potassium carbonate, a substance is obtained which is liquid, compact, opaque, yellow like wax, and crystalline." In the assays given in the table, the quantity of flux was increased in proportion to the percentage of potash added. As this alone might have had a bad effect on the result, a number of assays were made in which the percentage of potash was the same as in Nos. 64 to 68, the total amount of flux, however, not exceeding 30 grammes. They were all very unsatisfactory. Over fifteen minutes was required for the fusion, and there was no regularity in the results. To counteract this, assays were made, to which charcoal had been added to strengthen somewhat the weakened action of the diluted fluxes, placing, however, fluxes free from charcoal in the bottom of the crucibles. The buttons gave higher results, but the irregularity in their weights was the same as before. It would appear, therefore, that the potassium cyanide needs to be practically free from potassium carbonate; 9.09 per cent. of the latter showing already a loss of 1.34 per cent. in the result obtained.

* Berthier, *Traité des Essais*, 1847, vol. ii., p. 459.

† *Op. cit.*, vol. i., p. 445.

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(9.) The effect of sodium carbonate was tried in the same way. Although this alkali is not common with potassium cyanide, it was thought advisable to test it as, when potassium cyanide is used in blowpipe-experiments, it is generally mixed with an equal weight of sodium carbonate.

The mixtures were not so readily fusible as the corresponding potassium charges, and required fifteen minutes instead of ten as before. They were, however, just as thin, when once melted. When cold, the slags were similar to the potassium slags, only more vitreous and harder. The bluish-gray shades of color were missing; the slags were white, but had no luster whatever; the buttons were, on the whole, the same as above, but showed bead-like excrescences.

TABLE XV.

No. of Assay.	Charge.				Resulting Tin.	
	Ore.	KCy.	Na_2Co_3.			
	Grms.	Grms.	Grms.	Per cent. in Mixture.	Grms.	Per cent.
69	5	30	3	9.09	3.315	66.30
70	5	30	6	16.66	3.280	65.60
71	5	30	9	23.07	3.270	65.40
72	5	30	12	28.57	3.110	62.20
73	5	30	15	33.33	3.055	61.10
74	5	30	18	37.50	3.025	60.50
75	5	30	21	41.17	2.910	58.20

Table XV., it will be seen, contains no columns for tin in slags or crucible and the percentage of tin extracted is smaller than with the potassium salt, the inevitable conclusion being that an admixture of soda has a still more deleterious effect on the result than one of potash.

(10.) The influence of potassium sulphate is a very bad one. Bloxam* states that, in the presence of potassium sulphate, stannous and stannic sulphides are formed; the former, if the proportion of sulphate be small, the latter, if sufficiently large, the stannous sulphide remaining undissolved as a black powder if the slag be treated

* *Journal of the Chemical Society*, New Series, vol. iii., 1865, p. 97.

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with water, the stannic sulphide going into solution and being precipitated as yellow sulphide upon acidulating.

In the experiments, different amounts of potassium sulphate were added to 30 grammes of potassium cyanide and two-thirds of the new flux mixed with the ore, one-third being placed in the bottom of the naked crucible. The time required for tranquil fusion was ten minutes.

The charges fused readily and the crucibles were not attacked; the surface of all the slags was smooth, the color deepening with the percentage of potassium sulphate from light-red, with a dark rim, to black. The fracture was at first similar to that of pure cyanide, and grew finer until it became stony; the color, at first very light peach-blossom, became darker till it reached, with the last assay, a dark flesh-color. The hardness and toughness of the slags also increased with the percentage of potassium sulphate.

TABLE XVI.

No. of Assay.	Charge.				Resulting Tin.	
	Ore.	K Cy.	K_2SO_4.			
	Grms.	Grms.	Grms.	Per cent. in Mixture.	Grms.	Per cent.
76	5	30	0.3	0.99	3.185	63.70
77	5	30	0.6	1.96	No Button.	
78	5	30	1.0	3.22	3.060	61.20
79	5	30	2.0	6.25	2.525	50.50
80	5	30	3.0	9.09	2.490	49.80
81	5	30	4.0	11.76	2.435	48.70
82	5	30	5.0	14.28	2.250	45.00

The results obtained show a gradual decrease in the percentage of tin. One per cent. of potassium sulphate causes already a loss of 4.14 per cent. tin. In assay No. 76, some stannous sulphide was present; it decreased in No. 77, and was absent in No. 78, which, when acidulated, gave a copious precipitate of stannic sulphide. The deleterious influence of potassium sulphate, which was first shown by Bloxam, is thus verified by a series of systematic figures.

(11.) Various grades of the commercial potassium cyanide were

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obtained and tested. The grade of the cyanide can be judged from its general appearance, the pure being coarsely crystalline, soft, and snow-white. The lowest grade—"for mining purposes"—is reached when the structure becomes stony, and strongly resists pulverization; the color is a decided gray, resulting from finely divided iron.

In each experiment, 20 grammes of the special brand of cyanide were mixed with 5 grammes of ore, 5 grammes being reserved for the bottom of the crucible, and 5 grammes for a cover. The average time required for tranquil fusion was ten minutes.

TABLE XVII.

No. of Assay.	Granular Purified. Resulting Tin.		No. of Assay.	Pure for Gold Platers. Resulting Tin.		No. of Assay.	Common Fused. Resulting Tin.		No. of Assay.	Common for Mining Purposes. Resulting Tin.	
	Grms	Pr. ct.		Grms	Pr. ct.		Grms	Pr. ct.		Grms	Pr. ct.
83	3.225	64.50	86	3.370	67.40	89	3.220	64.40	92	3.115	62.30
84	3.225	64.50	87	3.355	67.10	90	2.990	59.80	93	3.010	60.20
85	3.100	62.00	88	3.340	66.80	91	2.905	58.10	94	2.900	58.00
Average	3.183	63.66	Average	3.355	67.10	Average	3.038	60.76	Average	3.008	60.16

a. "Granular Purified,"—86 to 90 per cent. KCy with some KO Cy and K_2CO_3,—$1.25 per pound.

The upper slag did not have the beautiful crystalline structure, nor the strong luster of the pure cyanide. It was white, but showed a bluish tinge near the top. The average result is 4.18 per cent. too low, and the greatest discrepancy between the single results is 2.50 per cent.

b. "Pure for Gold Platers,"—92 per cent. KCy, 5 per cent. KOCy and 3 per cent. other salts,—$1.25 per pound.

The general characteristics are same as those of Nos. 83 to 85, but the results are higher and more uniform.

c. "Common Fused,"—85 per cent. KCy,—$0.55 per pound.

The upper slag has a stony fracture and is of a dull-white color. The results are very low and very irregular.

d. "Common for Mining Purposes,"—65 per cent. KCy,—$0.50 per pound.

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The surface of the slag is smooth, has a dark-blue color with gray and yellow streaks passing through it. The fracture is stony, the slag is hard and tough, and is flesh-colored. The results are as bad as those from Nos. 89 to 91.

It will be seen that of the different grades, only the brand "Pure for Gold Platers" produces results—0.74 per cent. too low—which are sufficiently accurate for every-day purposes. As these brands represent each a certain definite grade of a particular firm, it will be seen that the assayer is obliged to test each separately, making the ordinary chemically pure potassium cyanide—98 per cent. KCy and 2 per cent. KOCy—his standard.

Class A, b.

This class embraces the bronze-assay of Winkler, and the different iron-assays.

4. Winkler's method* is as follows: 5 grammes of ore are mixed with 5 grammes of cupric oxide and charged in a clay crucible; on top are placed 15 grammes of black flux with 1.25 grammes of borax glass, then a salt cover, and finally a large piece of charcoal. The crucible is heated in a muffle or a pot-furnace, after boiling has ceased, for three-quarters of an hour to an hour at a bright-red heat, finishing the assay at almost a white heat. The result is a white, brittle, tin-copper alloy. With every tin-assay, an assay of 5 grammes of cupric oxide with the same charge is made, and the weight of the resulting button of copper, subtracted from the bronze-button, gives the weight of the tin contained in the ore.

This assay is really a modification of the German method. Its advantage is that no prills of tin (or, rather, particles of bronze) are found suspended in the slag. All the metallic oxides that have been reduced are found collected in one button. It is claimed that more tin is recovered by this method than by the regular German method, and that the buttons do not differ more than from 0.4 to 0.5 per cent. If this could be substantiated the method would be a serious rival to the cyanide assay.

In the experiments carried out the directions given above were closely followed, only the black flux substitute (2 potassium carbonate and 1 flour) was used, as in the regular German method, instead of the black flux proper.

* *Berg-und Hüttenmännische Zeitung*, 1864, p. 17; also Kerl, *op. cit.*, p. 483, and Balling, *Die Probirkunde*, p. 391.

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The charges were readily fusible, liquid, and showed a smooth surface when cool. The crucibles were not attacked, and no particles of bronze were discernible in the slags. As in the German assay, the upper salt slag and the lower borax slag have to be distinguished. The salt slag separates readily from the lower slag, has a crystalline structure, is brittle, has a pearly luster, and is opaque. In the centre of every salt slag is a small cavity, into which white, fern-like crystals protrude. The white base of the crystals is encircled by a greenly-tinged shell, and this by the bulk of the slag, which is of a lighter purple color than that of the German assay. The reason is probably that less carbon is present in the charge, no charcoal having been added. The vitreous borax slag is hard, opaque, and greenish-black. The oval bronze button is smooth,

TABLE XVIII.

No. of Assay.	Resulting			
	Bronze.	Copper.	Tin.	
	Grms.	Grms.	Grms.	Per cent.
95	7.200	3.825	3.375	67.50
96	7.170	3.805	3.365	67.30
97	7.110	3.785	3.325	66.50
98	7.105	3.785	3.320	66.40
99	7.040	3.820	3.220	64.40
100	7.020	3.770	3.250	65.00
101	7.000	3.770	3.230	64 60
Average	7.092	3.794	3.298	65.96
102	7.185	3.920	3.265	65.30
103	7.140	3.920	3.220	64.40
Average	7.162	3.920	2.2425	64.85

crystalline, brittle, white, and the more rounded copper button has threadlike lines of crystals on the surface, and is of a beautiful rose color. Both separate well from their slags.

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Table XVIII. shows that too much has been claimed for this method, both as to output of tin and accuracy of results. The total average, Nos. 95 to 101 (65.96 per cent. tin), is 1.88 per cent. too low, and while in some cases excellent results are obtained (Nos. 95 and 96), in others, under the same conditions, an assay can be 3.24 per cent. too low. It was thought that perhaps by increasing the amount of reducing agent, better results could be obtained. Thus 1 gramme of charcoal was mixed with the 5 grammes of ore and 5 grammes of cupric oxide, and then the usual charge given. The results (Nos. 102 and 103) show only an increase of weight in the copper recovered. Chalk-lined crucibles were tried, as in the German method. But, while no essential particular could be observed in which the assay differed from that in the naked crucible, the results given in Table XIX. show that the discrepancy in the resulting weights of tin is enormous, while the output of copper is higher and very even.

TABLE XIX.

No. of Assay.	Resulting			
	Bronze.	Copper.	Tin.	
	Grms.	Grms.	Grms.	Per cent.
104	6.850	3.815	3.035	60.70
105	6.660	3.815	2.845	56.90
106	6.760	3.810	2.950	59.00
107	6.250	3.810	2.440	48.80

As a modification, therefore, the chalk-lined crucible must here also be left out of consideration.

The reason that the results with the Winkler method vary so much is that the presence of copper adds a new difficulty to the assay. This can be seen from the tabulated results obtained in reducing cupric oxide for the tin assays carried out above.

In all the assays precipitated chemically pure cupric oxide was used. If reduced completely, 5 grammes of cupric oxide ought to give 3.993 grammes of copper. In the naked crucible we obtain as an average 3.801 grammes, and a discrepancy of 0.055 grammes between the highest and lowest results, which, if expressed in percentage, gives 1.1 per cent. difference. In the chalk-lined crucible there is

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a better output in copper; if extra reducing agents be added, a still better result is obtained, even in the naked crucible, but not a correspondingly better reduction of stannic oxide. It would seem,

TABLE XX.

Grammes of Copper Resulting in		
Naked Crucible with		Chalk-lined Crucible with Regular Charge.
Regular Charge.	Regular Charge and one gramme Charcoal.	
3.825	3.920	3.815
3.820	.....	3.810
3.805	.....	.....
3.785	.....	.....
3.770	.....	.....
Aver. 3.801	3.920	3.8125

therefore, that the amount of tin reduced will vary according to the amount of copper carried off in the slag.

In the assays with potassium cyanide it was seen that the presence of sulphates had a very bad effect on the result. As copper has a very great affinity for sulphur, the results of this bronze assay may also have been influenced by it. Several assays were made, omitting the salt and increasing the amount of potassium carbonate. The weight of the copper buttons was slightly higher; *e.g.*, 3.850 grammes, but the reduction of the tin was very imperfect, as it did not all alloy with the copper and collect in one button of bronze; prills of bright malleable tin were found in the slag, showing that to obtain a slag of sufficient fluidity for all reduced metal to collect in one button, the addition of salt is necessary. By reference to the German assay it will be seen that the presence of salt containing sulphates does not produce the same bad effect on tin as in the cyanide assay.

The Winkler method is given in a modified way by Ricketts:*

* *Notes on Assaying*, 1886, p. 89; quoting from Mitchell, *Manual of Assaying*, p. 411 (probably fourth edition, as it has been omitted in the fifth edition of 1881).

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Ore, 10 grammes; cupric oxide, 10 grammes; black flux substitute, 30 grammes; argol, 2 grammes; borax glass, 5 grammes, are well mixed, charged in a chalk-lined crucible, a salt cover and charcoal are added, and the charge heated gradually to a white heat for an hour. He advises separating tin and copper in the wet way. In this modified method it will be seen that an additional reducing agent—argol—has been added, and that flux and ore are intimately mixed. Winkler makes a point of not mixing ore and flux, as he wishes to avoid the strong boiling of the charge, whereby particles of alloy might adhere to the sides of the crucible and escape being collected in the button.

Three bronze assays were carried out and one copper assay, giving the following results:

TABLE XXI.

No. of Assay.	Resulting			
	Bronze.	Copper.	Tin.	
	Grms.	Grms.	Grms.	Per cent.
108	13.685	7.360	6.325	63.25
109	13.660	7.360	6.300	63.00
110	13.630	7.360	6.270	62.70
Average.	13.658	7.360	6.298	62.98

They show no advantage over Winkler's original method. Less tin is recovered and more copper is scorified, and a treatment of the alloy in the wet way would be apparently a loss of time.

5. *Iron Assays.*—Two sets of assays were made, the one recommended by Ricketts* for tin-ores containing silica only, the other by Mitchell,† for ores containing silica and for tin slags.

The method is based on the fact discovered by Berthier‡ and already alluded to, viz.: that iron completely precipitates tin from its combination with silica as metal, if added in sufficient quantity; if not, the stannic oxide is reduced only to stannous oxide, the iron

* *Notes on Assaying*, 1886, p. 89.

† *Manual of Assaying*, 1881, p. 483.

‡ *Traité des Essais*, 1847, vol. ii., pp. 459, 460, 484.

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itself being at the same time oxidized to ferrous oxide. Any excess of iron will alloy with the resulting metallic tin.

(1.) Ricketts gives the formula: Ore, 10 grammes; hematite, 3 to 8 grammes; potassium cyanide, 40 grammes, and says: "Mix and charge in a charcoal-lined crucible, cover with cyanide, and then with charcoal, lute and heat strongly from one-half to one hour, remove, tap carefully, cool, and break. If the tin be in small buttons, collect by washing with water to separate the charcoal, dry, and weigh. Treat the button as an alloy of tin and iron."

The result of the experiment was a porous grayish-black slag, through which small buttons of alloy were finely disseminated, and in the bottom of the crucible a button. Button and larger prills were picked out, the crucible lining crushed, the pulp screened, the scales and the siftings, obtained from panning the pulp, weighed with button and larger prills. The weights of the alloys were 8.355 and 8.320 grammes, which shows a close agreement if the amount of iron is the same in both cases. This assay was not further pursued, as there are more simple methods for obtaining accurate results. The silica of ores "containing silica only" can be treated in a much more simple way. It can be removed by washing, and the pure black tin assayed with potassium cyanide, or, if too great a loss of cassiterite is feared, the silica can be removed by hydrofluoric acid, and the resulting pure black tin then assayed. This method of assay has therefore little value for the assayer, except that it substantiates an interesting fact, viz., that iron carries down all the tin.

(2.) Mitchell* mixes 400 grains of ore with 200 grains of ferric oxide, 100 grains of fluorspar, and 100 grains of charcoal powder, charges the mixture in a covered crucible, keeps it at a dull-red heat for half an hour, and then at a white heat for another half hour.

In the experiments the above quantities were reduced to a basis of 10 grammes of ore. These resulted in a hard, rough, bluish-black shell, with bronze-colored stains. The inside was filled with fine charcoal, no button or any prills being visible. Apparently the ore and the hematite had united to form a slag. The method was not further investigated. If tin slags are to be assayed, it is simple to decompose the finely pulverized slag with sodium or potassium disulphate and then treat with acidulated water, when stannic oxide and silica will be left. The tin in this mixture can then be readily determined.

* *Manual of Assaying*, 1881, p. 484.

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(3.) One more experiment remains to be mentioned here, *i.e.*, the fusing of black tin with potassium ferrocyanide and potassium cyanide. Bloxam* states that by using this reagent all the tin can be obtained as an alloy. The idea that led to the experiments was that if a weighed amount of dried ferrocyanide were fused with the ore, the buttons obtained from a number of assays would all have the same weight. If all the iron set free by the decomposition of the ferrocyanide alloyed with the tin, the percentage of tin could be readily calculated.

In the experiments, 5 grammes of ore were mixed with 5 grammes of dried potassium ferrocyanide and 5 grammes of potassium cyanide. For the bottom of the crucible and for the cover 5 grammes of potassium cyanide were reserved. The results of the experiments were negative. The alloy was partly disseminated through the lower gray slag, which was porous and tough, and had partly collected in a button. The weights of several 5-gramme assays did not agree within 1 gramme, showing that all the iron of the ferrocyanide had not united with the tin, but that part had remained suspended in the lower slag.

Class B.

The second group of assays comprises the Cornish methods, which are only for determining the tin obtained by treating the ores on a large scale. That the results of such assays must be wrong is conceded. How closely they may agree when repeated in the same way by the same person is not stated.

6. Mitchell† and Ricketts‡ give the following method "for pure tin oxide;" 400 grains (Mitchell) or 10 grammes (Ricketts) are placed in a charcoal-lined crucible, to which a cover is luted. The crucible is heated gradually for 25 minutes, finishing the operation at a white heat.

For the experiments, 25 grammes of ore were treated as described. There resulted in the bottom of the crucible a dull button of tin, while on the walls and bottom were numberless bright prills of tin. Button and prills were free from iron. The charcoal lining was pulverized and made to pass a 60-mesh sieve. The button, the scales on the sieve, and the siftings resulting from the pulp that had been panned were weighed together. The weights of two assays were:

* *Journal of the Chemical Society*, New Series, vol. iii., 1865.

† *Op. cit.*, p. 480.

‡ *Op. cit.*, p. 88.

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18.120 grammes = 72.5 per cent. tin.
16.150 " = 64.6 " "

The chemical analysis shows only 67.84 per cent. of tin. The reason for this strange result is to be sought in the panning of the fine pulp. When all the charcoal has been washed off there remains in the pan a heavy black residue. It is almost impossible to see what part of this is fine tin and what heavy slag, as they scarcely differ in appearance. It is certain that with the result 72.5 per cent., slag still remained with the tin, and it is not at all sure that it was all removed with the result 64.7 per cent. Perhaps after long acquaintance with uniform ores some approximately reliable estimates might be obtained, but as far as exact results go the method appears to the writer a very doubtful one.

7. Another method is given by Mitchell:* 2 ounces of ore are mixed with a small quantity of culm and projected into a red-hot naked crucible; some fluorspar is added if necessary, and after a quarter of an hour's fusion the tin is poured. The prills in the slag are recovered by pounding and vanning.

In the experiments, 50 grammes of ore were mixed with 12 grammes of finely pulverized anthracite, shot into the red-hot crucible and heated for half an hour, a little fluorspar having been added. No fusion was apparent. The crucible was removed and broken when cold. It contained a black, porous, fritted mass, full of prills of tin, and on the bottom a button. The entire lower part of the crucible was crushed, screened, and panned, and all the recovered tin weighed together. The weights were:

34.10 grammes = 68.20 per cent. tin.
27.38 " = 54.70 " "

The same difficulty as above in separating the heavy black slag from the fine tin causes the irregularity.

8. In connection with the Cornish methods it may be mentioned here that it has been the writer's custom, in melting out metallic tin from different grades of black tin, to mix the black tin with 5 per cent. of lime, 5 per cent. of fluorspar, and 12 to 15 per cent. of charcoal or anthracite, the latter requiring a higher temperature but giving cleaner slags. With pure concentrates obtained from ore occurring in quartz no special difficulties have been encountered, so that the entire contents of a plumbago crucible, weighing about 10

* *Op. cit.*, pp. 480 and 481.

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pounds, can be readily poured. With impure concentrates the results of such fusions have always been somewhat pasty slags, and additions of lime or fluorspar were never of much avail. The only remedy has been to add salt, which makes the slags less pasty. The main point in all these fusions is always to have the crucible nearly filled. If the first charge has begun to fuse, a second one is added. Such a fusion takes from one to two hours. Unfortunately no records have been kept of the amount of white tin contained in the black, which would give the amounts saved.

In reviewing the preceding experiments, it is striking that so many different methods and modifications should still be recommended when a few simple ones are quite sufficient. The experiments made show clearly that there are only two methods giving entire satisfactory results. These are the German method, as exemplified by assays Nos. 1 to 6, and the method of fusion with potassium cyanide, as from assays Nos. 35 to 38; or, still better, if sufficient material is available, Nos. 39 to 42.

In the following investigations concerning the influence the different minerals associated with the black tin have on the results of these assays, we shall see to which of the two methods the final preference is to be awarded.

V. Influence of Associated Minerals on the Assay.

The minerals that are liable to remain with the black tin after the ore has been washed, roasted and treated with nitro-hydrochloric acid, are quartz, feldspar, mica, tourmaline, garnet, and columbite (tantalite). To test their influence, the minerals were ground fine enough to pass through a 60-mesh sieve, and any particles of iron coming from the implements used in grinding were removed by the magnet. The minerals were mixed with the black tin in the different proportions shown in the subjoined tables, and the effects studied, which they have on the two best methods of assay—the German method and the method of fusing with potassium cyanide. The charges were so regulated that the flux in the German assay was always equal to three times the quantity of the ore plus the mineral, and in the cyanide assay to six times the quantity. Four parts of cyanide were mixed with the ore, one part served as bottom for the crucible and one part as cover. In the German assay, the amount of borax glass was increased in proportion to the percentage of mineral added and that of salt decreased. The fusions were made

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in the usual way; the time for the cyanide-assay had to be increased to not less than fifteen minutes before a tranquil fusion was obtained.

(1.) The quartz used was the ordinary compact, colorless variety.

TABLE XXII.

No. of Assay.	CHARGE.							RESULTING TIN.	
	Ore.	Quartz.		Charcoal.	Bl. Flux Substit'e	Borax Glass.	Potass. Cyanide.		
	Grms.	Grms.	Per ct.	Grms.	Grms.	Grms.	Grms.	Grms.	Per cent.
111	5	1	16.66	1	18	1	.....	3.280	65.60
112	5	2	28.57	1	21	1	.....	3.100	62.00
113	5	3	37.50	1	24	.....	.....	3.035	60.70
114	5	4	44.44	1	27	.....	.....	2.945	58.90
115	5	5	50.00	1	30	.....	.....	2.565	51.30
116	10	2.5	20.00	.....	Bl. Flux. 50	.....	.....	5.2	52.00
117	10	6.6	40.00	.....	66.4	.....	.....	4.3	43.00
118	10	10.0	50.00	.....	80	.....	.....	2.8	28.00
119	10	15.0	60.00	.....	100	.....	.....	1.0	10.00
120	10	30.0	75.00	.....	160	.....	.....	0.0	00.00
121	5	1	16.66	.....	.....	.....	24	3.285	65.70
122	5	2	28.57	.....	.....	.....	28	3.215	64.30
123	5	3	37.50	.....	.....	.....	32	3.195	63.90

The amount of salt slag found on top of the borax slag decreased in proportion to the percentage of quartz added, showing that part of it had entered into combination with the borax slag. In assay No. 111 it was ¼ inch thick, in No. 115 only ⅛ inch. Its structure was not so coarsely crystalline as usual. It grew finer, and the brittle slag became harder with increase of silica; the luster remained the same; the color varied from gray to bluish-black. The borax slags increased in quantity as quartz was added, their vitreous character being somewhat impaired by the silica and also their brit-

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tleness; the luster remained uniform; the usual olive-green color changed somewhat and became brownish. The buttons separated well from their slags, had the usual appearance, and were free from iron. The results obtained in Nos. 111 to 115 showed a gradual decrease in the amount of tin recovered. This is, however, less than has been generally supposed from the figures of Berthier, Nos. 116 to 120, which are usually quoted.* Berthier,† as the writer understands it, mixes his siliceous ore with four times the amount of black flux. With the German method of charging, the black tin, with 67.84 per cent. of tin mixed with 50 per cent. of silica gave 51.30 per cent. of metallic tin; Berthier's black tin containing 70 per cent. of tin, with the same addition of silica, but mixed with the flux, gave only 28 per cent. of tin or 24.8 per cent. less than if charged as in the German method. This shows the disadvantage of mixing ore with basic flux.

In the cyanide-assays, Nos. 121 to 123, the slags showed uniformly a very coarsely crystalline structure and had a strong luster. They were as opaque as usual, and became harder with the increase of silica. In No. 121 only the upper rim was parrot-green. This spreads in No. 122 so as to cover the entire surface; it is $\frac{1}{4}$ inch thick at the circumference and $\frac{1}{8}$ inch in the center; in No. 123, the thickness is uniformly $\frac{1}{4}$ inch. The appearance of the buttons is the same as if no silica had been added. The results show a decrease in the amount of tin recovered, but the loss is not so great as with the German assay.

(2.) The feldspar used was the compact albite that occurs so generally in the tin lodes; it has a very perfect cleavage, shows a pearly to a silky luster, and is white with a tinge of gray.

Salt- and borax-slags of the German method, Nos. 124 to 128, are similar to those obtained when quartz was added to the ore. The buttons, however, are surrounded by a slag that is more stony than the borax-slag above, and almost black. It adheres strongly to the buttons, and is liable to retain particles of metal. All the buttons were dark and dull; they were brittle, hard to cut, and contained iron, its presence being caused by the infusible character of the albite; the reduction was prolonged and the ferric oxide of the cassiterite reduced before fusion took place, and alloyed with the metallic tin resulting from the more readily reducible stannic oxide.

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* Mitchell, *op. cit.*, p. 483; Ricketts, *op. cit.*, p. 87.

† *Traité des Essais*, 1847, vol. ii., p. 485.

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By examining the results, Nos. 124 to 128, it will be seen that, although all the buttons are ferruginous, the weight of the four last buttons is too low, showing that tin had also entered the slag, not enough iron being present to precipitate all the tin. The slagging is assisted by the intimate contact of the refractory feldspar and the

TABLE XXIII.

No. of Assay.	CHARGE.							RESULTING			
	Ore.	Albite.		Charcoal.	Bl. Flux Substitute.	Borax Glass.	Potass. Cyanide.	Tin.		Alloy.	
	Grms.	Grms.	P. cent.	Grms.	Grms.	Grms.	Grms.	Grms.	P. c't.	Grms.	P. Ct.
124	5	1	16.66	1	18	1	.....	.....	.....	3.475	69.50
125	5	2	28.57	1	21	1	.....	.....	.....	3.265	65.20
126	5	3	37.50	1	24	1.25	.....	.....	.....	3.010	60.20
127	5	4	44.44	1	27	1.50	.....	.....	.....	2.680	53.60
128	5	5	50.00	1	30	2	.....	.....	.....	3.615	52.30
129	5	1	16.66	.....	.....	.....	24	3.285	65.70	.....	.....
130	5	2	28.57	.....	.....	.....	28	3.275	65.50	.....	.....
131	5	3	37.50	.....	.....	.....	32	3.265	65.30	.....	.....

cassiterite. Owing to the character of the charge, stannic oxide, or even metallic tin, is liable to be slagged by the alkali carbonate and stannous oxide by the silica.

The influence on potassium cyanide is seen by Nos. 129 to 131 of the table. The upper white slag of pure cyanide with cyanate adheres to a lower porous slag of greenish-gray color, which encloses the buttons. These are white, bright, malleable, easily cut and free from iron. The amount of tin lost, even if 37.50 per cent. of albite is added to the ore, is only a little over 2 per cent. It is strange that there should be so little difference between the three assays which contain such varying amounts of albite. The only explanation seems to be that, at the low temperature at which the assay is carried out, the feldspar is not decomposed, but remains suspended in the lower slag, which receives its greenish tinge from the iron of the cassiterite and its gray from the feldspar mixed with it. If the lower slag is boiled in water, it falls to pieces, leaving a residue that

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looks very much like the original pulverized albite after it has been moistened, except that it has a slight tinge of green, resulting from the iron.

(3.) The mica used was a pure white muscovite.

TABLE XXIV.

No. of Assay.	CHARGE.							RESULTING			
	Ore.	Muscovite.		Charcoal.	Black Flux Substitute.	Borax Glass.	Potas. Cy'nide	Tin.		Alloy.	
	Grms.	Grms.	Pr. ct.	Grms.	Grms.	Grms.	Per ct.	Grms.	Pr. ct.	Grms.	Pr. ct.
132	5	1	16.66	1	18	1	.....	.....	.....	3.190	63.80
133	5	2	28.57	1	21	1	.....	.....	.....	3.250	65.00
134	5	3	37.50	1	24	1.5	.....	.....	.....	3.200	64.00
135	5	4	44.44	1	27	1.5	.....	.....	.....	3.605	72.10
136	5	5	50.00	1	30	2	.....	.....	.....	.....	.....
137	5	1	16.66	.....	.....	.....	24	3.260	65.20	.....	.....
138	5	2	28.57	.....	.....	.....	28	3.235	64.70	.....	.....
139	5	3	37.50	.....	.....	.....	32	3.165	63.30	.....	.....

The general behavior of the assays according to the German method, Nos. 132 to 136, is the same as those with feldspar, the exception being that with 50 per cent. of mica, only an incomplete fusion could be obtained. In the bottom of the crucible was a porous, fritted, dark mass, no tin being visible; it was covered by a semi-fused, scoriaceous, spongy mass of a brown color. The buttons had the same appearance and properties as those from the feldspar-mixture. The results show a greater loss of tin and a stronger reduction of iron than above, the reason being that muscovite is more refractory in the assay than albite.

In the assays with potassium cyanide, Nos. 137 to 139, the behavior of mica is similar to that of feldspar, the mica appearing also here more refractory.

(4.) The tourmaline pulverized for the experiments was the large black crystal which occurs so often with the tin ore.

In the German method, Nos. 140 to 144, the upper crystalline slag is parrot-green, the lower slag is slightly vitreous, but quickly be-

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comes stony with the increase of tourmaline. The buttons have all a gray surface; they are dull, brittle, hard to cut, and ferruginous. These properties increase with the addition of tourmaline. In assay No. 144 no complete fusion could be obtained and no button or particles of alloy were visible in the porous fritted dark mass. The

TABLE XXV.

No. of Assay.	CHARGE.							RESULTING			
	Ore.	Tourmaline.		Charcoal.	Black Flux Substitute.	Borax Glass.	Potass. Cy'nide	Tin.		Alloy.	
	Grms.	Grms.	Pr. ct.	Grms.	Grms.	Grms.	Grms.	Grms.	Pr. ct.	Grms.	Pr. ct.
140	5	1	16.66	1	18	1	.....	.....	.....	3.470	69.40
141	5	2	28.57	1	21	1	.....	.....	.....	3.300	66.00
142	5	3	37.50	1	24	1.5	.....	.....	.....	3.250	65.00
143	5	4	44.44	1	27	1.5	.....	.....	.....	2.295	45.90
144	5	5	50.00	1	30	2	.....	.....	.....	.....	.....
145	5	1	16.66	.....	.....	.....	24	3.285	65.70	.....	.....
146	5	2	28.57	.....	.....	.....	28	3.250	55.00	.....	.....
147	5	3	37.50	.....	.....	.....	32	3.235	64.70	.....	.....

results show a loss of tin increasing with the addition of tourmaline, and an increased amount of iron is taken up by the tin.

With potassium cyanide tourmaline has no effect that is essentially different from that of previous assays, as the slags show about the same appearance, and the buttons the same properties and nearly the same weight.

(5.) The garnet occurring in the tin deposits is the iron-garnet (almandite). It has a brownish-red to fine deep-red color, and is very rich in iron. This appears in its high specific gravity, 4.149, and from the fact that, after being heated by itself in a crucible to a bright-red heat, it shows magnetic properties. For the experiments transparent crystals of a fine deep-red color were pulverized.

From this table it will be seen that no borax was added to the charges of the German assay, Nos. 148 to 152, as the garnet with the addition of a small amount of base (lime or alkali) forms a liquid slag. The salt slags were less crystalline than usual, but otherwise

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the same with the exception of the color, which varied from light to dark red. The borax slags lost their vitreous and brittle character by the addition of garnet, became stony and tough, and changed from subtranslucent to opaque. The color, a dull olive-green, became lighter, the last slag—No. 152—being peacock-green. The

TABLE XXVI.

No. of Assay.	CHARGE.							RESULTING ALLOY.	
	Ore.	Garnet.		Char-coal.	Bl. Flux Sub.	Borax Glass.	Potass. Cyanide		
	Grms.	Grms.	Pr. ct.	Grms.	Grms.	Grms.	Grms.	Grms.	Per ct.
148	5	1	16.66	1	18	.....	.....	3.685	73.70
149	5	2	28.57	1	21	.....	.....	3.720	74.40
150	5	3	37.50	1	24	.....	.....	3.695	73.90
151	5	4	44.44	1	27	.....	.....	3.930	78.60
152	5	5	50.00	1	30	.....	.....	4.125	82.50
153	5	1	16.66	.....	.....	.....	24	3.365	67.30
154	5	2	28.57	.....	.....	.....	28	3.435	68.70
155	5	3	37.50	.....	.....	.....	32	3.490	69.89

buttons were all a dull dark-gray, hard to cut, brittle, and rich in iron. The weights obtained in the first four assays, Nos. 148 to 151, are strikingly close. This would seem to show that about the same amount of iron had been present in all of them, forming an alloy of nearly the same composition, the rest of the iron having remained in the slag. The uniformly well-melted charges and the beautiful manner in which the one button separates from the slag above, show that if tin is to be recovered from siliceous ore as an alloy, a basic silicate of iron is the best, because it gives up the necessary iron for the tin and also retains sufficient in combination to form with the alkali of the charge a readily fusible, brittle slag.

In the cyanide assays the compact white upper slag decreases, and the porous bluish-green lower slag increases as garnet is added to the charge. In No. 153 the proportions were half and half, in No. 155 one-quarter white and three-quarters gray. The buttons are all ferruginous, and show the usual characteristics. The figures prove that, although iron has been taken up, tin has gone into the slag.

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(6.) The columbite found with the tin ores has an iron-black color and a streak that varies from a very dark brown to a pure black. The pure mineral has the following composition:*

	Per Cent.
Ta_2O_5,	18.20
Nb_2O_5,	64.09
FeO,	11.21
MnO,	7.07
SnO_2,	0.10
CaO,	0.21
Total,	100.88

The specific gravity varies between 5.89 and 6.12. The columbite used for the experiments contained a small proportion of albite.

TABLE XXVII.

No. of Assay.	CHARGE.							RESULTING			
	Ore.	Columbite.		Charcoal.	Black Flux Substitute.	Borax Glass.	Potass. Cy'nide	Tin.		Alloy.	
	Grms.	Grms.	Pr. ct.	Grms.	Grms.	Grms.	Grms.	Grms.	Pr. ct.	Grms.	Pr. ct.
156	5	1	16.66	1	18	1	.....	.....	.....	3.400	68.00
157	5	2	28.57	1	21	1	.....	.....	.....	3.605	72.10
158	5	3	37.50	1	24	1.5	.....	.....	.....	3.775	75.50
159	5	4	44.44	1	27	1.5	.....	.....	.....	3.820	76.40
160	5	5	50.00	1	30	2	.....	.....	.....	3.875	77.50
161	5	1	16.66	.....	.....	.....	24	3.355	67.10	.....	.....
162	5	2	28.57	.....	.....	.....	28	3.370	67.40	.....	.....
163	5	3	37.50	.....	.....	.....	32	3.300	66.00	.....	.....

In the German assays, Nos. 156 to 160, the salt slags were stony and dull; their color, at first a brownish-green, gradually lost the brown and became finally parrot-green. The borax slags were vitreous, brittle, and had a resinous luster, the color changing from a light to a dark olive-green. The buttons were all ferruginous; their color, at first a dark-gray, became black. The results show that the amount of iron reduced increased with the percentage of columbite

* *Trans.*, xvii., 593.

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added to the charge. Also, that if tin-ore containing columbite be treated on a large scale, the columbite will probably behave in a way similar to the wolframite of Cornwall ores, where the iron that alloys with the tin is the principal cause of trouble, and not the tungsten of the mineral, which, according to Berthier* may be present in considerable quantities.

In the experiments with potassium cyanide the upper slags showed the usual properties, the lower slags were very porous, slightly glassy, and had a dark blue-gray color. The porosity increased with the percentage of columbite added. The buttons were white, bright, malleable, easily cut, and contained very little iron; from their appearance they might have been supposed to be free from iron, until especially examined for this metal. If pure cassiterite and pure columbite are ground together and fused, thus intimately mixed, with potassium cyanide, the iron and manganese of the columbite combine with the tin, leaving a yellowish slag which contains the columbic and tantalic acids.†

By comparing the effect produced by the associated minerals in the two methods of assay, we see that the cyanide-assay must be accorded the preference, although, if the amount of associated mineral be small, the German method will give entirely satisfactory results. In concentration works, where several assays have to be made daily, the question of cost must be considered, and the cheap German assay will probably be preferred to the more expensive one, especially if the assay is only made in order that the manager may know how pure his concentrates are. In questions connected with the buying and selling of ores or concentrates, the assay with potassium cyanide is better, because the associated minerals have comparatively little influence on it.

VI. Preparation of the Ore for the Assay.

It will be seen from the foregoing that the influence of the associated minerals on the result of the assay is very considerable. It becomes, therefore, necessary to have as pure a black tin as possible for the dry assay. When it is remembered that tin-ores contain only from ¾ to 3 per cent. of cassiterite, their preparation for the assay is seen to be a very important part of the work.

The separation of the cassiterite from the gangue is universally

* *Traité des Essais*, 1847, vol. ii., p. 472.

† Drs. Carpenter and Headden, *Trans.*, xvii., 633.

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accomplished by crushing and washing. The resulting concentrates are then purified by means which vary according to circumstances. From the great difference in specific gravity between the cassiterite (6.4 to 7.1), and the non-metallic associated minerals (not over 4.1), it would appear at first sight that the separation ought to be very easy, but this, for various reasons, is far from being the case. Special precautions are needed if the loss of black tin is to be reduced to a minimum. The metalliferous minerals have too great a specific gravity (*e.g.*, arsenopyrite 6 to 6.4) to be removed by washing; roasting and acids have to be used. The columbite (tantalite) finally, whose specific gravity varies from 5.89 to 6.12, remains with the black tin, as it is not attacked by ordinary acids at all.

The implements used in assay laboratories for pulverizing larger quantities of ore are a small hand-crusher and a pulverizing plate. The ore is usually crushed, sampled down, and then pulverized on the rubbing-plate or in a mortar. Experience has proved that the common method of grinding the whole sample to a uniform size is not suited to the tin-ores of the Black Hills. The main reason for this is the preponderance of mica. Any one who has tried to pulverize mica, even if previously heated, will appreciate the difficulty. Happily very little cassiterite occurs in the mica proper, which if ground for a short time only on the rubbing-plate will not show any black tin or other mineral. The pulverization must be carried on with the subsequent washing in view. The first rule in concentration is that a good preliminary sizing should precede the separation by specific gravity. The second is that the ore should be coarse to prevent loss in washing. The latter point is strongly emphasized in the present case by the fact that the brittle cassiterite will be ground to a slime, while other minerals in the ore are only converted into sands. The writer's practice has been to screen while the ore is being pulverized, crushing and screening alternately with a graduated series of sieves, Nos. 20, 40, 60 and 80. The ore is crushed until all the minerals, with the exception of the mica, are fine enough to pass a 20-mesh sieve; the mica is then examined for any minerals adhering to it and these are removed by grinding, and screened. The different screenings are now washed separately in the gold-pan and the tailings panned over. The concentrates obtained by the separate washings are put together, dried, and passed through a 60-mesh sieve and the washing is repeated, the result being pure concentrates and tailings. These are panned over if any cassiterite appears.

The writer has done all his washing of tin-ore in the gold-pan;

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but any other apparatus, be it the vanning shovel used in Cornwall, or the "Sichertrog" of the Saxon or Bohemian miner, will, with the necessary practice, give the same result. In panning, it is found very difficult to separate the cassiterite from the garnet without losing much tin, although there is a great difference in their specific gravities, viz., 6.4 to 7.1 and 4.1. It is almost impossible to effect a complete separation, as can be seen from the analysis of the average sample, where the insoluble residue, 9.39 per cent. is principally garnet. This is on account of the different way in which the minerals break. Dana* says that both cassiterite and garnet have an uneven, subconchoidal fracture, and that both are brittle; they also differ little in hardness (6 to 7 *vs.* 6.5 to 7.5). But the cassiterite crushes into splinters and the garnet into a granular form. It is probably this difference that causes the difficulty in separating the two minerals; the preponderating particles of cassiterite form an irregular angular network and prevent the heavy garnet from being carried off by the water with the other minerals that have a lower specific gravity. If garnets are present in any large quantities, it will be necessary to adopt other means to remove them. Dana† says that all garnets, except ouvarovite, are decomposed with hydrochloric acid after ignition. That this may be complete, the mineral has to be ground very fine. It will be remembered that the black tin used for the experiments was roasted, chilled, and treated with nitro-hydrochloric acid, and the garnets were not entirely removed. If the garnet is decomposed, the removal of the gelatinous silica, either by washing or by hydrofluoric acid, is simple.

The black tin obtained by washing still retains sulphides and arsenides. Berthier‡ recommended boiling with nitro-hydrochloric acid and burning off any sulphur that might have separated. Plattner§ objects to this, as the sulphur would not all be eliminated. He reverses the process, roasting first and then treating with nitro-hydrochloric acid. Following Plattner, the black tin is roasted on a dish in a muffle, and, to assist the decomposition of the garnet, chilled in cold water after roasting; it is then boiled in a porcelain dish with nitro-hydrochloric acid, which will also remove particles of metallic iron coming from the implements used in grinding;

* *System of Mineralogy*, 5th ed., New York, 1880, pp. 158 and 265.

† *Op. cit.*, p. 270.

‡ *Traité des Essais*, 1847, vol. ii., pp. 486, 487.

§ *Blowpipe Analysis*, edited by Richter and translated by Cornwall, New York, 1835, p. 466.

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when all the iron is dissolved the solution is poured off, and any light particles washed away. If necessary, the concentrates can be transferred to a platinum dish and treated with hydrofluoric acid. This, however, will rarely be required.

The roasting seems to have a favorable effect on the cassiterite by making it more friable and more readily reducible. The unroasted cassiterite of the Black Hills is subtranslucent, brown to black, and very hard to pulverize; the roasted mineral is opaque, of a more or less reddish-brown, and easily pulverized. The change that takes place in the reducibility of the ore is shown by the following experiments:

TABLE XXVIII.

No. of Assay.	Name of Mine.	Per cent. of Tin from	
		Raw Ore.	Roasted Ore.
168	Occidental.....	48.90	49.00
169	Glendale.....	65.30	65.60
170	First Find.....	65.30	65.80
171	Similias.....	75.00	75.00

The results show that in some cases (No. 171) no difference exists between raw and roasted concentrates, while in others it may amount to 0.5 per cent.

If the preliminary treatment of a tin-ore is carried out in the manner described, there will be a minimum of loss in black tin.

VII. CONCLUSION.

From this examination of the different methods of assaying tin-ores in the dry way, it becomes apparent that, if ordinary precautions are taken, the necessary treatment, including the preliminary preparation of the ore, is a comparatively simple matter; neither is the influence of the associated minerals so great, at least with the cyanide assay, as is generally supposed. Lastly, the results obtained show that the dry assay of purified black tin gives more accurate results than that of any other base metal if the few correct methods, none of them requiring special skill, are followed and ordinary care is given to the work.

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PROGRESS OF METALLURGICAL SCIENCE IN THE WEST.

BY RICHARD PEARCE, ARGO, COLO.

(Presidential Address at the Colorado Meeting, June, 1889.)

I AM deeply sensible of the honor you have conferred on me in electing me your president for this year. It is difficult to understand why I have merited such distinction at your hands, except that I may possibly have helped, in some small way, in the development of metallurgical practice in the West. I fear, however, that I shall fall far short of the expectations you may have in regard to my fitness for such a position.

The responsibility of preparing an address for this occasion has caused me considerable thought and anxiety. Unlike my predecessors in office, many of whom have dealt in their presidential addresses with subjects covering a wide field, which they could survey with the confidence born of their own extensive knowledge, I have been confined by my experience, during the last twenty-five years, almost wholly to that portion of metallurgical science which bears especially on the treatment of gold and silver. Yet perhaps some sort of general review of the progress of this branch of metallurgy in the West, besides being the contribution I feel myself best qualified to offer, may prove acceptable to many of the members of the Institute.

Some few of the gentlemen present will remember the crude condition of metallurgical practice as it was conducted in Colorado only twenty-five years ago. My own personal experience covers a period of eighteen years; and during this comparatively short time I have seen the most astonishing changes in the conduct of metallurgical works. In all probability the marked progress which this country has exhibited could hardly be reached anywhere else, and this result has been achieved mainly by the activity and enterprise so characteristic of the men of the West.

The first little successful attempt at smelting made at Black Hawk, in 1867, by Ex-Senator N. P. Hill, representing the Boston and Colorado Smelting Company, proved a great boon to the miners,

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who were almost overpowered with the difficulties of ore-reduction. His works were erected at a time when mining was threatened with collapse; and they afforded a ready market for ores which failed to yield their gold by the old-fashioned stamp-mill process. The whole operation of smelting, as conducted at that time, was a very simple one, and, in these days, would be regarded as a process involving very little metallurgical skill, rewarded by what would appear to be very large profits. But when we consider the difficulties which even the most experienced metallurgist had to contend with, such as the scarcity and inferior quality of building and refractory materials, the high price of labor and fuel, and the heavy freight charges which the product had to bear in its transportation to Swansea, Wales, we are not surprised that the rates for treatment which prevailed were so high.

The system adopted by the Boston and Colorado Smelting Company at that time, and so successfully conducted by Prof. Beeger, the father of smelting in the West, consisted in the concentration of the gold, silver and copper into a rich matte by two operations, viz.: roasting, and smelting in reverberatory furnaces. The degree of concentration was about eight or ten tons into one. The further treatment of the matte, for the extraction of the silver and gold, was given in Swansea, Wales.

These facts are not merely points of technical reminiscence; they could scarcely be overlooked by any historian of Colorado, for the stimulating effects of this little establishment at Black Hawk had a marked influence on the subsequent development and importance of this great State.

I shall have occasion, in the course of this address, to refer again in a general way to this process as it is now conducted at the works at Argo, which were established in 1878.

For a period of ten years, from 1870 to 1880, other metallurgical systems were represented in Colorado, in the shape of amalgamation-works, or mills, of a different type from the gold-mills of Gilpin county. These were established in the silver-districts of Georgetown and Caribou, and one or two other localities, and were, on the whole, fairly successful; at the present time, however, owing to the low cost of smelting, there is not a single silver-amalgamation mill in successful operation in this State.

The treatment of gold- and silver-ores by smelting with lead was practiced to some extent in the early days of mining in Colorado; but, owing to the scarcity of the latter metal, lead-smelting can

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hardly be said to have been successful. When I visited Georgetown in 1871, I was credibly informed that the "Brown Mining and Smelting Company," one of the old pioneer mining companies of Clear Creek county, had been obliged to import from Chicago, in the form of pig-lead, and at a cost for freight alone of $130 per ton, the lead required for the reduction of silver-ores; and, in the summer of 1874, the old Swansea works, near Denver, used granulated lead for the reduction of silver-ores, in reverberatory furnaces of very small capacity. Various attempts were made in different parts of the State to smelt ores by the lead system, but I think I am correct in saying that scarcely one of those establishments was conducted with any financial success.

The year 1877 was marked by the great discoveries in Leadville; and in the following year lead-smelting began in earnest. At the outset everything was necessarily very crude; and the first attempts at smelting were conducted without much regard to scientific principles. The rich carbonate of lead, however, proved to be easy of reduction; and sufficient local skill was found to conduct the business with profit.

As a result of keen competition, improved methods were soon adopted. Under the skilful guidance of such men as Grant, Eilers, Fohr, Meyer, Hahn, and others, the business soon settled down to a practice based on sound principles; and to-day we have monuments of the skill and enterprise of these gentlemen in Denver, Pueblo, and Leadville, which, I think, will compare favorably with any other lead-smelting establishments in the world.

The superior skill which has been brought to bear on smelting is the result of a more intelligent comprehension of the scientific principles involved; and a great deal of valuable knowledge has been gained from the experience of a few years of practical work. Changes have been made in the form and size of the furnaces; the proper composition of the slag has been determined; and a great many devices and improvements have been adopted to keep pace with the vigorous competition which has existed in Colorado since the era when lead-smelting began. To this, mainly, we may ascribe the superior advantages which Colorado offers as a market for ores. These advantages are felt in a great many districts outside of the State; for a large proportion of the ore smelted comes into the Colorado market from other States and Territories.

The following table, which I have prepared from data collected by a friend, will show the commercial advantages which the miner

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has experienced by the progressive development of the smelting industry. For the sake of comparison, I have selected ores which have no special value as fluxes, or as aids to smelting, and will consider them merely in relation to their intrinsic value, and endeavor to show the returns which the miner gets now, as compared with the figures of eighteen years ago. In other words, the net percentage value of an ore to the miner to-day is compared with the value for successive periods from 1871. The value of the silver has been figured from $1.29 per ounce in 1871, down to 93 cents in 1889, and for the years prior to the resumption of specie payment, the premium on gold is taken into consideration. The slight falling off in 1874 and 1875 was due to the depression following the financial panic in the fall of 1873. For a time, silver-ores were rather a drug on the market, and prices consequently fell off.

Table Showing Percentage of the Total Value of Ores Paid to Miners each Year during a Period of Eighteen Years.

Place.	Year.	Contents in Silver per ton.	Percentage of total value paid to the miner.
Black Hawk	1871	100 ozs.	65.
	1872	"	65.
	1873	"	65.5
	1874	"	53.6
	1875	"	60.
	1876	"	67.2
	1877	"	64.3
	1878	"	65.
Argo	1879	"	70.
	1880	"	74.
	1881	"	74.
	1882	"	76.
	1883	"	76.5
	1884	"	81.
	1885	"	77.
	1886	"	80.
	1887	"	80.
	1888	"	82.
	1889	"	84.

The difference between the maximum and minimum is 30.4 per cent.

I have avoided making any figures to show the changes in value of gold-ores from year to year, but without going into details, I may state that a Gilpin county gold-ore which would net the miner

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53 per cent. of its value in 1870, would now yield him 80 per cent., a difference of 27 per cent. and, on a somewhat lower grade of ore, the difference is 33 per cent.

For what are commonly called tailings (or concentrated pyrites from the stamp-mills) the net percentage value to the miners in 1874 was 24 per cent.; at the present time it would amount to 76 per cent., a difference, in favor of the miner, of 52 per cent.

A pyritous ore from the "Robinson mine" in 1882, assaying 50 ounces of silver per ton, would net the miner, at Argo, 55.4 per cent. of its value; at the present time the same ore would net 78 per cent. of its value.

The increase in value in these particular ores is mainly due to their value as fluxes, containing, as they do, a large percentage of iron and very little silica.

The difference in the value of Leadville ores, extending over a period of ten years, is even more marked than the changes I have shown in regard to silver-ores in the older mining districts. The changes in Leadville have been largely effected by the demand for lead, and certain basic material necessary for successful smelting. The different ores command varying prices, depending on their composition, and the fortunate mine owner who possesses an ore which fills all the requirements of the smelter receives a premium, whilst his unlucky neighbor, whose ores are of a different character, suffers heavy discounts.

In 1875 silver was worth about $1.40 per ounce in currency. Its present value is something over 92 cents, or 34.28 per cent. less than in 1875. Taking the output of the Colorado mines alone for 1888 in round numbers at 18,000,000 ounces, this reduction in price amounts to the sum of $8,600,000, which is considerably beyond the total value of the gold and silver product of the State before the development of the Leadville mines.

It will be seen from these figures that if it were not for the great efforts which have been made from time to time to cheapen the cost of smelting, silver mining here would have received its death-blow long ere this.

The decline in silver has brought the net value of the ore to the miner down to about the same as it was thirteen years ago, but the cost of mining has been largely reduced by improved facilities for transportation, by cheaper labor and by cheaper materials, enabling him to sell at a profit ores that formerly were worthless. These same elements have, of course, helped the smelter to a still greater degree, and there is yet room for further improvement.

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Through the kindness of Mr. Guiterman, I am able to present a statement showing the quantity of ore smelted by the smelting works in Colorado for the year 1888: Total ore smelted, 596,594 tons; this includes 72,147 tons from other States and Territories, leaving a total tonnage of ores produced in Colorado of 524,447, or 88 per cent.

In the year 1877 the total smelting capacity of Colorado probably did not exceed 20,000 tons, or about 3½ per cent. of the amount treated last year.

The present capacity of lead blast-furnaces, of modern construction, is from 60 to 65 tons of material per day, exclusive of fuel. This represents, on an average, from 40 to 45 tons of ore per day.

As you will all have opportunities of seeing the operation of smelting in blast-furnaces, and judging for yourselves of the magnitude and importance of this branch, I will not attempt to draw attention, at this time, to the nature of the improvements referred to. There are gentlemen present who are more familiar with these points than I am. I would like, however, briefly to make a comparison of the conditions in the two systems of smelting, namely: In a blast-furnace and in a reverberatory furnace. Although the latter is not adapted to the treatment of lead-ores, yet it would appear to have some merits when used for smelting ores containing copper, in lieu of lead, as a base for collecting the precious metals.

For convenience I have tabulated the peculiar conditions of the two systems:

	Blast-Furnace System.	Reverberatory System.
Atmosphere	Reducing.	Oxidizing.
Fuel	Coke, charcoal.	Coal, wood.
Degree of Concentration	Low: 6 or 7 in 1.	High: 20 in 1 before separation of silver.
Per cent. of base metal required to collect the precious metals	15 per cent. of lead.	3 per cent. of copper.
Lime as flux	Necessary.	Unnecessary.
Calcining pyritiferous ores	Should be complete.	Not necessarily complete.

The old-fashioned reverberatory furnace, in use ten years ago, with a capacity of ten tons per day, and a ratio of fuel to ore of 1 to 1¼, would stand no chance whatever in competition with the present style of blast-furnace. But certain changes in the construc-

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tion of the reverberatory within the last two years have increased the capacity to about 25 to 27 tons per day, with a ratio of fuel to ore of 1 to 3. This, perhaps, is the highest record reached by reverberatory smelting, except some work said to have been accomplished in Montana by introducing red-hot ore into the furnace direct from the roasting cylinders.

The system adopted by the Boston and Colorado Smelting Company has certain features which are, perhaps, somewhat novel. Although copper is used, in lieu of lead, as a means of collecting the silver and gold contained in the ores, it can hardly be said to represent copper smelting, for this metal plays only a small part as regards the value of the output. Although the general principles of the process would entitle it to be called the "Swansea system," I think the changes and improvements which have, from time to time, been introduced show some features distinctly American.

The old works of the Boston and Colorado Smelting Company, at Black Hawk, which were so well described by Dr. Egleston in 1875,* have disappeared, and the method in use at Argo to-day has been so far changed as to be scarcely recognizable as the old Boston and Colorado process, except so far as the general principles involved. Nothing has been published relating to these works since Dr. Egleston's paper appeared, and I may, perhaps, be permitted to give a short sketch of the system adopted without going into details.

The present capacity of the plant is 200 tons of ore per day. These include a great variety of copper, silver and gold-ores from Colorado as well as from other western States and Territories. About 12¾ per cent. of the ores treated comes from districts outside of the State. These include Montana, Utah, Nevada, California, Arizona, New Mexico, and old Mexico. In composition the ores differ very widely, made up as they are of an infinite variety of minerals. A few typical examples are:

Silver- and gold-bearing pyritous ores from different districts of Colorado.

Barytic silver-ores from Aspen.

Siliceous silver- and gold-ores from Colorado.

Pyritous and arsenical concentrates from Colorado and Nevada.

Arsenical copper-ores from Utah.

Copper mattes from Montana, and from the lead-smelting establishments of Colorado.

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* *Trans.*, iv., 276.

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The general average contents of the ore as smelted from day to day are: Silver 40 to 60 ounces, and gold ½ to 1 ounce per ton, and 2 to 3 per cent. of copper. About 90 per cent. of the ore treated contains no copper, the other 10 per cent. furnishing the copper to make up the general average to about 2 to 3 per cent.

Fifty per cent., or 100 tons, of the ore has to be roasted per day. This operation is conducted partly in furnaces and partly in kilns, and in neither is it carried on to a finish, as sulphur is always allowed to remain to the extent of from 5 to 7 per cent. The remaining 100 tons are smelted without any preliminary roasting.

I need not, at this time, describe the changes which occur in roasting. All this has been well done before by other members of the Institute, and I only take this opportunity to refer particularly to the excellent little work on copper smelting by H. M. Howe. I will, however, notice very briefly a point connected with the elimination of arsenic in this stage of the process.

I have found by experience that it is not an easy matter to get rid of arsenic from an ordinary arsenical copper-ore by roasting, and in order to test the efficiency of coal added in the latter stage of roasting for the elimination of arsenic, as recommended in certain works on copper smelting, I have made the following experiments:

Experiment No. 1: On an ore consisting of chalcopyrite, bornite and enargite, and containing 3.26 per cent. of arsenic, gave, on roasting to a "dead roast," a product which contained 2.5 per cent. of arsenic; loss, 23.31 per cent.

Experiment No. 2, on the same ore: Sample roasted dead and, whilst still hot, mixed with 10 per cent. of fine coal. Roasting continued until coal was consumed. The roasted ore was found to contain 2.19 per cent. of arsenic; loss, 32.82 per cent.

Experiment No. 3: The same ore mixed with 10 per cent. of fine coal and the whole well roasted. Product contained 2.36 per cent. of arsenic; loss, 27.60 per cent.

Experiment No. 4: Sample of ore was well roasted and then mixed with 30 per cent. of coal. It was then submitted to a strong heat in a covered vessel to exclude air and, finally, it was roasted with the admission of air until the coal was consumed. The roasted ore was found to contain 2.36 per cent. of arsenic; loss, 27.60 per cent. The mean of these results show a difference in favor of coal of only 6.02 per cent.

All the above experiments were made in a muffle on finely pulverized ore, and the conditions were as perfect as it is possible to obtain.

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A sample taken from the same ore, which had been roasted for twenty-four hours in a large calciner, gave 1.59 per cent. of arsenic; loss, 51.22 per cent. This result is no more than might be expected, for, as has been clearly pointed out before, the conditions which are most perfect for oxidation are not the best, in practice, for the elimination of arsenic from copper-ores. This fact is explained fully by H. M. Howe and others.

The result of my experiments, as given above, would indicate that the reducing action of coal, or any other carbonaceous matter, on arseniate of copper, is not so active as is generally supposed. Indeed, it is a question in my mind whether there is any action of importance, or whether reduction can take place except at the melting point of arseniate of copper, which is beyond the temperature necessary for roasting. It is, of course, possible that arseniate of copper is reduced to arsenide of copper at a roasting temperature in contact with coal without producing any sensible loss of arsenic.

I will now return to the description of the process at Argo.

The roasted ore is mixed with others, principally siliceous, and the mixture is so arranged that, when smelted, it shall yield a slag containing, as near as possible, 40 per cent. of silica and a first matte (ore-metal), which assays 40 per cent. of copper, 400 ounces of silver and 6 ounces of gold per ton. The capacity of each furnace is about 25 tons of ore per day, the total burden being 27 tons, on account of rich slags that are added from other operations.

The degree of concentration in this first smelting is about 13 into 1.

The composition of the slags, so far as the bases are concerned, is subject to many changes. The silica ranges from about 38 to 42 per cent., and no lime or flux is used other than what may be contained in the ores.

Examples of Typical Slags.

	I.	II.	III.	IV.	V.	VI.	VII.
Silica	40.96	39.80	41.62	37.42	39.42	36.52	41.73
Protoxide of iron	28.30	29.40	28.26	33.00	31.41	30.68	33.08*
Alumina	2.96	4.35	4.84	3.63	.....	1.29	.....
Oxide of Manganese	7.16	5.53	5.03	3.72	4.53	4.46	3.96
Oxide of Zinc	9.10	11.03	7.87	11.47	13.46	20.08	9.72
Lime	7 08	7.60	5.59	5.01	5.52	3.57	6.02
Magnesia	0.76	0.94	0.58	trace	.....	.....	1.34
Copper	0.39	0.62	trace	0.42	0.40	.....	.....
Lead	0.42	0.24	3.31	3.78	.....	.....	.....

* Including alumina.

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III. and IV. are slags from smelting ores containing lead.

VI. is slag produced from ores containing much zinc.

The ratio of the oxygen in the base to that in the silica is 1 to 1.5, and the slag would be classed as a sesquisilicate.

Since these slags all show distinct magnetic properties, we may assume that a portion of the iron exists in the form of magnetic oxide. The oxidizing conditions of the laboratory of the furnace will account for this peculiarity in the slag, as, at the end of the operation, the reducing action of the sulphur and sulphurous acid ceases, to a certain degree, to be active.

These slags were all produced some five years ago, but they represent nearly the composition of the slags made at this time, except that the latter contain barium as an additional base, since Aspen ores have come into the market.

The matte which is produced in this first operation of smelting consists mainly of copper, iron and sulphur, but when, as is sometimes the case, lead is present in the ore, a portion of this metal replaces the iron in the matte. This frequently occurred in former years, before care was taken in the assortment of the ores. An example of this copper-lead matte is given in the following analysis:

	First matte (ore metal.)	Probable composition of first matte.	
Copper	32.03%	Cu_2S	39.85%
Lead	26.60	Pb S	30.47
Iron	9.50	Fe_2S_3	17.51
Zinc	4.29	Zn S	6.36
Silver	1.92	Ag_2S	2.18
Gold	0.05	Au	0.05
Sulphur	21.70		
Insoluble residue and traces of other metals	3 58	Insol. res. and traces of other metals,	3.58
	99.67		100.00

This is merely given to show to what extent lead may enter into the composition of the matte without seriously interfering with the subsequent treatment. It may be of interest to note the results of some assays from two samples of matte from one large tap, representing about 7 tons. When broken, the mass showed a section of about 12 inches, the upper part of which varied considerably from the lower. The upper portion was of a very close texture, and the lower crystalline and vesicular.

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	Copper. Per cent.	Lead. Per cent.	Silver. Oz. per ton.	Gold. Oz. per ton.
The upper portion contained, . .	38.6	18.9	543.	8.4
The lower portion contained, . .	31.	37.1	760.	18.

The ratios between the quantities of lead, silver, and gold in the two portions are approximately as follows:

	Lead.	Silver.	Gold.
Upper portion, ...	1	1	1
Lower portion, ...	2	1.4	2

One great objection to the existence of lead in large quantities in such mattes is, that a compound of lead, copper, iron, and sulphur is formed, still richer in lead than the original matte which, from its extreme fusibility, finds its way into the hearth of the furnace. It is often found in lustrous crystalline plates, having somewhat the structure of spiegeleisen, and corresponding in composition very closely to the following formula:

$$5\,(Cu_2S) + 4\,(PbS) + Fe_2S_3.$$

An analysis of this substance gave:

	Per cent.
Copper, ...	32.60
Lead, ...	40.52
Iron, ...	5.02
Silver, ...	2.56
Sulphur, ...	19.97
	100.40

I may state here that this first matte always contains a certain amount of lead, but the quantity rarely exceeds 10 per cent.

I pass on to the next stage in the process, which includes the roasting and concentration of the ore-metal, or first matte, to "white metal" containing 60 per cent. of copper.

The charge in this operation of smelting consists of a mixture of two-thirds of roasted metal and one-third raw metal, together with rich siliceous silver-ore in quantities sufficient to prevent corrosion of the furnace by the iron present in the matte.

The concentration furnace in which this operation is performed stands some 3 feet above the level of the ore furnace, in order that the rich slag from the former may, when skimmed, flow directly into

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the ore furnace, where it is cleaned and reduced to the condition of ordinary ore-furnace slag. This effects a very great saving in the re-smelting of these rich slags.

The white metal from this concentration-smelting contains, on an average, from 700 to 800 ounces of silver and 10 ounces of gold per ton and 60 per cent. of copper. Each ton of this matte represents from 20 to 21 tons of ore.

This white metal is now ready for the extraction of the silver, which comprises the following operations:

Rough roasting.

Fine grinding.

Fine roasting for sulphate of silver.

Leaching, and the precipitation of the silver.

Refining and melting of the cement silver.

The metal, crushed so as to pass through a screen of 6 meshes to the linear inch, is roasted for twenty-four hours. The bulk of the sulphur is driven off in this operation, and oxides of copper are formed, the mass still containing sufficient sulphur for the subsequent operation of roasting for sulphate of silver. After cooling, the roasted material is ground fine in a Chilian mill and passed through a 60-mesh screen. It is then ready for the fine roasting and conversion of the silver into a sulphate.

Roasting for Sulphate of Silver.

This operation, based on the principles first pointed out and carried into practice by the eminent Ziervogel, is, perhaps, one of the most interesting and most beautiful of all the metallurgical processes ever discovered. The chemical changes involved in the operation, so far as they are understood, have been described again and again, and I will only briefly refer here to the principal stages in the roasting process.

It must be understood that the process is a delicate one, and requires to be watched with great care in order to arrive at good results. The whole operation extends over a period of about five hours from the time the charge is placed in the furnace until its withdrawal.

The *first stage* is conducted at a low temperature, and lasts about one and a half hours. The finely-ground material begins to glow from the oxidation of any copper disulphide and copper dioxide that may be present.

Second Stage.—Slightly increased temperature for one and a half

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hours. Material increases in volume, and assumes a porous, spongy appearance from the formation of cupric sulphate.

Third Stage.—Temperature further increased for one hour, or until sample treated with water shows the silver to be what is termed "out," which means that it has been changed into a sulphate. The presence, at this stage, of a quantity of copper dioxide would, during the operation of leaching with water, reduce the sulphate of silver to metallic silver, the latter showing itself in beautiful spangles.

Fourth Stage.—Charge is vigorously stirred and turned over to present fresh surfaces for oxidization, the temperature being kept as in the third stage. This stage is purely for the purpose of completely oxidizing any cuprous oxide, so that no reducing action shall occur when the material is leached.

The presence of foreign elements in the matte very materially influences the results in this operation. Arsenic, antimony, and bismuth form insoluble compounds with silver. Bismuth is found to be most injurious, and gives more trouble than all the other metals together. An insoluble sulphate of bismuth and silver is probably formed, which increases the richness of the residues in silver.

In the absence of large quantities of these metals, or with what might be considered fair average matte, the residues will contain, after leaching, about 40 ounces of silver per ton; but as these residues are subjected to a further treatment, the question of a few ounces of silver, more or less, makes very little difference in the final result, as will be explained.

The leaching of the roasted matte is effected in tubs, and nothing but hot water is required. The solution, charged with sulphate of silver, is allowed to flow over copper plates for the precipitation of the silver. This process of precipitation requires little attention. The solution enters tank No. 1, charged with silver, and leaves tank No. 2, charged with copper, and containing no trace of silver. This copper solution, in its turn, flows over old scrap iron for the precipitation of the copper.

In the precipitation of the silver a certain amount of copper is found mixed with the silver in the form of cuprous oxide and of small scales and scraps of metallic copper, and a process of refining is necessary previous to melting. This copper is removed by prolonged boiling with water containing a small quantity of sulphuric acid, into which air is injected by means of a small jet of steam. Sulphate of copper is formed, which is carefully washed out of the silver. The silver is then dried and melted into bars of an average fineness of 999.

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The residues, after the extraction of the silver, contain about 40 ounces of silver and 10 ounces of gold per ton, and 55 per cent. of copper in the form of cupric oxide. They also contain associated metals, such as lead and bismuth as sulphates, and iron as ferric oxide.

Treatment of the Residues.

The residues are mixed with the necessary quantities of pyritous ores containing gold, or low grade silver-bearing pyrites rich in sulphur, and with siliceous gold-ores, and are smelted direct for matte of 65 per cent. copper. This matte contains about 10 to 15 ounces of gold per ton and 80 ounces of silver. The slag from this operation is thrown away. It contains about the same proportion of iron and silica as the ore-slag.

Treatment of Matte for Rich Copper-Gold Alloy.

This process involves two operations, which may be described under the following heads: A combined roasting and smelting, and a refining of crude alloy.

The aim of this process is to concentrate the gold contained in the matte into a rich gold-silver-copper alloy, containing also the bulk of the impurities existing in the matte. This process resembles the old Swansea process for making "best selected copper," a description of which will be found in any of the standard works on copper-smelting. As practiced at Argo, it is as follows:

Twelve tons of matte, or residue metal, are placed in slabs on the hearth of a large reverberatory furnace. The openings of the furnace are closed, except two large ports, which are situated near the bridge, and are used for the supply of air for oxidization. The heat is gradually increased for a period of about seven hours, or until the whole mass assumes a semi-pasty condition. The ports are then closed, and the heat is still further increased until a point is reached, when the whole charge is in a perfectly molten condition. In the last stage of the melting a reaction takes place between the oxide and sulphide, and a certain amount of metallic copper is liberated. This metallic copper contains nearly all the gold, together with any foreign metals present in the matte, such as lead, arsenic, antimony, bismuth, etc. The charge is tapped into sand moulds, in the first few of which, nearest the taphole, there will be found, after cooling, under the enriched matte, plates of the above-mentioned impure copper, commonly known as metallic bottoms. The proportion of

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these bottoms to the whole tap of matte, called pimple metal, is about 1 to 15. The following analysis will give some idea of the composition of this crude alloy:

	Per cent.
Copper,	60.04
Lead,	33.61
Arsenic,	0.44
Bismuth,	0.40
Iron,	0.08
Zinc,	0.15
Gold,	0.54
Silver,	1.35
Sulphur,	1.68
	98.29

It must be understood that this analysis shows the composition of one specimen of such alloy only. The composition varies, of course, with the nature and quantity of the impurities present in the residue metal. These impurities, without a doubt, increase the solvent action of the copper on the gold. In other words, a copper matte containing little or none of the impurities mentioned, will not yield its gold so readily to the copper. The whole operation of roasting and smelting occupies about twelve hours. The crude bottoms contain 100 to 200 ounces of gold to the ton, and about 300 ounces of silver. The compound that bismuth forms with gold, when present in this crude alloy, has been referred to in my paper on "Certain Interesting Crystalline Alloys."* The matte, pimple metal, if the operation has been properly done, contains not more than 0.2 ounces of gold per ton (assays frequently show less than 0.1 ounce of gold), 90 ounces of silver, and 77 per cent. of copper. The roaster slag from this operation is smelted in the ore furnaces.

Treatment of the Crude Alloy.

The treatment of this alloy and the separation of the gold from the copper is a part of the process which, for certain business considerations, I am not at liberty to describe.

Treatment of the Pimple Metal or Finished Matte.

The finished matte, containing 77 per cent. of copper and 90 ounces of silver per ton, is crushed, roasted, ground fine, and again

Trans., xiii., 738.

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roasted to form sulphate of silver. In fact, it is treated precisely in the same manner as the rich silver matte described in the Ziervogel process. The only difference is, that the treatment is conducted in a separate set of furnaces, as the matte contains no gold. The residue, after leaching, consists almost wholly of oxide of copper, with an amount of silver not exceeding 10 ounces per ton. The oxide of copper is dried, packed in barrels and sold to the manufacturers of sulphate of copper, or reduced to black copper by melting with fine slack coal in a reverberatory furnace. This copper is refined in Massachusetts and disposed of in the eastern market in the form of ingot copper.

At this point let me refer for a moment to some important improvements which have been recently introduced in Montana in the treatment of copper-ores. The old, slow and costly method of roasting in calciners is being replaced, with great success, I learn, by very capacious revolving cylinders. In a country where fuel and labor are dear, a change of this nature is of immense advantage, when large quantities of ore have to be roasted before smelting. Transferring the hot ore direct to the smelting furnace, instead of allowing it to cool before charging, is also a very important gain, as it greatly increases the capacity of the furnace.

Whilst Colorado may be said to lead in its successful development of the lead industry, Montana has come to the front and has made a great advance in the treatment of copper-ores. One can form some idea of the magnitude of the copper-smelting enterprises in the neighborhood of Butte, when I state that the works of the Anaconda Company alone have a capacity sufficient to smelt 500 tons of ore per day, yielding about 100 tons of copper in the form of matte. And this would have been largely increased had it not been for the disastrous fire which, last March, destroyed some extensive additions.

The Anaconda works, which were first built some four years ago, were moulded after the Welsh pattern, with all the old-fashioned methods peculiar to the system which has been so long in vogue in South Wales. It was soon discovered that, with expensive labor and high-priced fuel, some radical change must be made. The works were consequently entirely remodeled in a style which, from their magnitude and the ingenuity shown in handling the materials, may be considered purely American.

Montana may also, I believe, be credited with the only application of the French system for Bessemerizing copper mattes in this

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country, and from the long period it has been in use, I presume that success has been assured.

I have, so far, limited my remarks to the progress which has been made in smelting. I cannot, however, lose sight of the advance which has been made in other branches of metallurgical work bearing on the extraction of silver, and I would take this opportunity of referring briefly to the "Russell Process."

It would appear that this process, which has been so elaborately worked out and so well described by the inventor and Mr. Stetefeldt,* is capable of being applied very generally where conditions are unfavorable for smelting, and we shall watch with great interest the practical results of the different establishments which have been started in a great many districts in the West and in Mexico.

The process appears to have a great future as a substitute for the old process of amalgamation, and will, I have no doubt, come more into competition with that particular system of ore treatment than with smelting.

I would suggest the desirability of adopting at the works, some simple refining operation for the treatment of the rich sulphide of silver which is produced by precipitation. This material is often too rich to be transported any very great distance at ordinary freight rates, and the tax which it has to suffer from heavy express charges adds very greatly to the cost of putting the silver into the market. This is a subject which, in my opinion, should receive the attention of those interested in this process so as to make it less dependent on smelters for a final disposition of the product.

I have, in this short address, endeavored to show the business advantages which the country has derived by the progress of metallurgical science in the West; I must not lose sight of the fact, however, that this progress is largely due to the interchange of thought among the members of this Institute.

A perusal of the volumes of the *Transactions* of the Institute will convince one of the prominence given to this branch of science.

There are, however, certain limits to the amount of information which we feel we ought to give to the public, and in this we are governed less by any selfish policy of our own than by the wishes of those gentlemen with whom we are associated in business. American engineers are probably less subject to restrictions of this kind than their brethren on the other side of the Atlantic—if we

* See *Trans.*, xiii., 47; xv., 355; and xvi., 362.

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may judge from the testimony of those gentlemen who, from time to time, have visited this country.

In Europe it is frequently the custom to guard with jealous care certain operations which really have no merit whatever, but are merely examples of imperfect and antiquated work. I fear this is the case to a very large extent in England, as I have had frequent opportunities of seeing, but this is not due so much to inefficiency on the part of the man in charge, as to the bigotry and self-superiority of the proprietor, or chief of affairs, whose sole idea of excellence is to run carefully in the footsteps of his great-grandfather, or some ancient relative, who got rich in the business because he couldn't help it.

Believe me, gentlemen, a man who knows his work well, and is not handicapped by conditions altogether beyond his control, need not fear competition, for in the race for progress he practically solves that great problem which prevails throughout the whole range of life, viz.: "The survival of the fittest."

AN OCCURRENCE OF COPPER GLANCE NORTH OF LAKE HURON, WITH NOTES ON THE STRUCTURE OF THE LOCALITY.

BY JAMES T. B. IVES, F.G.S., TORONTO, CANADA.

(Colorado Meeting, June, 1889.)

THE variety of copper-ore to which these notes refer is comparatively rare, and, so far as I am aware, has not been recorded hitherto as occurring in Ontario. Moreover, the rocks of this locality differ so much in nature and mode of occurrence from the ore-bearing rocks of the western United States, that a description of them may be interesting.

A line drawn upon a map of the region of the great lakes from the mouth of the Severn river at the Southern end of Georgian Bay, following the Canadian shore northwestward, and leaving Manitoulin island to the south, will describe the boundary between the Archæan rocks lying north of it and the Silurian rocks southward.

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The former are Lower Laurentian as far as the Indian village of Killarney, at which point their line of separation from the Huronian comes in, and continues westward, trending thence to Wahnapitae village, where the river of that name is crossed by the Canadian Pacific railway. In these Huronian beds occurs the Vermilion Company's gold-mine, in Dennison township, thirty miles northeast of the locality described in this paper; and ten miles farther, in the same direction, are the extensive copper- and nickel-mines of the Canadian Copper Company. Ninety miles west, along the coast, is the Bruce Mine, classical in the history of copper-mining, but now idle and possibly exhausted.

The Silurian rocks of Manitoulin island and the headland which approaches it from the south represent several successive formations. Those exposed along the southern shores of the island and peninsula are limestones of the Guelph formation. North of them is a broad belt of Niagara limestone, succeeded on the island by Hudson river beds, with patches of Utica shales and Trenton limestones, while along the northern shores are found Birdseye and Black River, and probably Chazy beds. All these are exposed in succession, a gentle uprise to the north having promoted the erosion of the northern beds. As a rule, the lines of outcrop are marked by escarpments, which form a conspicuous feature in the landscape, and almost make it possible to distinguish the succession of the formations in viewing the island from the deck of a steamboat at a distance of several miles. The southern dip of these beds is, however, so slight that it is not perceptible to the eye, even when considerable tracts are exposed to view. I have seen in various parts of the island floors of limestone flags, extending for probably one hundred paces from east to west, and somewhat less from north to south, apparently as level as though paved by man.

Just north of the eastern end of the Manitoulin island is a narrow peninsula, extending into the North Channel in a direction slightly south of west. To the south of it is another in the form of a tuning-fork of a similar size and trend. Upon the new admiralty chart the form and size of these peninsulas is more accurately shown than on ordinary maps. The northern one, with which we are here concerned, is about ten miles long, its greatest width being one mile. The bay which separates it from the peninsula south of it is nowhere more than a mile wide, narrowing at the outlet to one-eighth of a mile, and its length is seven miles. I draw attention to this parallelism of the peninsulas because it is characteristic of the

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distribution of such land as rises above water-level, whether as promontories or islands, and is doubtless connected with the geological structure of the country.

The Sault Ste. Marie steamboats stop at several villages on Manitoulin island, of which the one nearest to this peninsula is Little Current. Looking out thence toward these headlands, and the mainland generally, it is striking how very different their rocks are from those of the island itself, the appearance being that of a range of lofty hills, mostly of rock, so white in places and in certain lights as to give the impression that they are covered with snow. Twenty miles eastward from Little Current is a point half-way along the peninsula. As the boat approaches the shore, the difference in aspect becomes more conspicuous. Instead of the horizontal, usually pale buff limestones of the island, there are rounded hills of crystalline rock, either white or tinted blue, green, or brown approaching black. In leaving the island, however, the horizontal Silurian beds are not entirely left behind. They may be seen in isolated patches as beaches and shore-line sections, from two to ten feet in height. In narrow channels such beds may frequently be seen on one side, while on the other are crystalline rocks. I observed three varieties: a buff sandy (or rather calcareo-arenaceous) conglomerate, some of the pebbles in which are white quartz, jasper and chlorite schist; lithographic limestone, superimposed upon the former; and variegated shales. The latter occur on the peninsula, close to the deposit to be described, and also several miles northwest of that point. They rise at the water's edge to a height of six or eight feet, and consist of laminæ, averaging one inch in thickness, of three well-defined colors—brick-red, buff and copper-green—the colors usually alternating, but two being sometimes mottled on one of the tile-like laminæ. While this bed resembles the country-rock at Sault Ste. Marie, and is probably of the same age, its lamination is finer.

There is a definite succession of the two kinds of crystalline rock mentioned above. The white is quartzite, at places highly talcose, and frequently, at other points, passing into quartzose conglomerate by inclusion of pebbles. The dark rock is slate, likewise conglomeritic in some of its beds, and occasionally passing into chlorite schist. The rock where I moored my boat and pitched my tent, which consequently became very familiar to me, was of bright pink quartzose conglomerate, with partings of talc schist of pink and greenish tints. It was only on close scrutiny that I discovered the minute

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pebbles which the quartzite contained, and of which, on examination, I found it was almost made up in places.

These two varieties of rocks are distributed in bands running parallel with the length of the peninsula, about S. 80° W. There are eight or ten such bands, showing distinct lines of separation. Standing on such a line on an elevated spot, and looking east or west, the junction may be seen as far as the eye can reach, excepting at intervals where obscured by vegetation or lost in the hollows. The rocks have been sculptured into ridges and furrows parallel with the direction of the peninsula and the upturned edges of the beds. This sculpturing is doubtless due to inequality in the hardness of the strata constituting both the quartzite and the slate. It must not be supposed that the ridges are alternately of quartzite and slate; the junction between those rocks may occur along a ridge or a hollow, or on the slope between. The quartzite has certainly resisted erosion more than the slate, the bolder ridges being of the former rock, which also greatly exceeds the latter in quantity,—that is to say, in the relative thickness of the beds,—probably as three to one. They range from ten to several hundred feet in thickness. Besides the longitudinal ridges and hollows, there are minor transverse depressions which give the hills, as seen from the south or north, a serrated appearance.

Although my observations were specially directed to a particular locality, situated near the center of the peninsula, I may remark that in physical features this peninsula resembles the islands, promontories and mainland of the region generally, except in the absence of dioritic dykes, which are common in the neighborhood. At this central point the peninsula narrows to about a quarter of a mile, and the hills are lower than elsewhere, being fifty to two hundred feet high, whereas they are generally two or three hundred feet in height. Two of the transverse gullies here are almost level with the bays on either side; and a lakelet occupies part of the space between them, they being only a quarter of a mile apart. An old Indian portage along one of the gullies attests that the natives have realized the convenience of this natural gateway between the waters.

Here an interesting dislocation of the strata has taken place. Two beds of slate (A and B, in Fig. 1), which run parallel for a considerable distance with a band (C) of quartzite between them, come together, the quartzite thinning out. This is evidently due to faulting, which has caused a relative downthrow of one portion as

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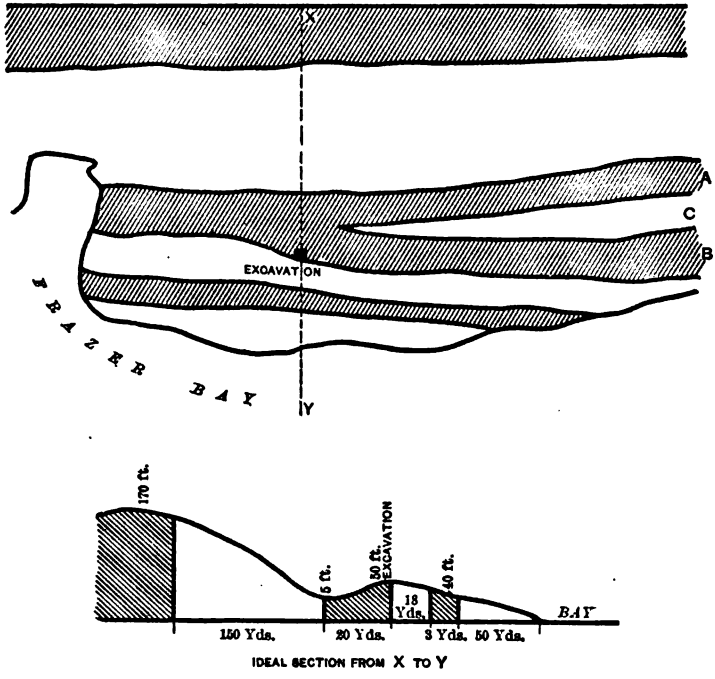
compared with the other, the plane of displacement being slightly oblique to the plane of bedding, and the movement more considerable at one end than at the other. Besides this unmistakable fault there is other evidence of disturbance, for the rocks are traversed in all directions by minute veins of quartz. The slate-bed at this point is impregnated with grains of hematite. The next ridge north of this is quartzite. It rises to a height of about one hundred and seventy feet, and at that end of it, adjacent to the western gully and an

FIG. 1.



Plan and Section.—Not drawn to Scale.

inlet (shown on the sketch), it also is charged with hematite. The rock appears to have been more or less completely converted into hematite. There is no defined separation of the ore-bed or gangue from the enclosing rock, the latter gradually passing into solid ore. An exception was noted at one point where an irregular band of chloritic schist was observed, from one to several inches in width, marking the line of separation. This hematitic impregnation imparts a reddish-brown color to quartzite and slate alike. Whether this deposit of iron occupies the hollow separating the two ridges

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cannot be ascertained without some amount of work, as the rocks are masked with soil and vegetation. It may do so; and the hollow probably owes its existence to that fact, the presence of the ore rendering the rock more susceptible to erosion.

A deposit of copper glance is found in the slate bed where the latter is charged with iron. It occurs in the form of strings, approximately at right angles to the plane of faulting above referred to. The strings are generally one or two inches thick, and may be followed downwards almost vertically. Where exposed near the surface, the ore is coated with the green carbonate of copper, giving a brilliant appearance to the deposit. The matrix rock, besides being impregnated with grains of hematite and coated in the fissures with a putty of red ochre, contains small quantities of magnetite. When broken up the slate-rock is found to have lost its normal appearance and to have assumed that of a breccia of quartz, hornblende and chlorite, a transformation which is due probably to a segregation of the constituent minerals when exposed to the great heat accompanying the faulting and fissuring described above. This rock is often much stained with the green carbonate, presenting the appearance of an impure malachite. Average samples of the ore yielded by assay 56 per cent. of metallic copper, while picked specimens gave 72 per cent. The rock yielded 1 oz. 4 dwts. of silver, besides about 2 dwts. of gold, to the ton. The quartz of the veins also yielded gold in small quantity.

The bed of slate in which the deposit occurs is found, when traced east and west across the gullies, to contain, where exposed at the surface, quartz veins of reddish color, carrying the form of copper-ore common in that region, viz., chalcopyrite, or copper pyrites.

How far this copper deposit may prove profitable commercially, I cannot tell. In its present undeveloped condition it is interesting as indicating some of the processes supposed by Prof. Blake to have attended the formation of the ores of Copper Basin, Arizona,* chalcopyrite being probably the mineral from which both the chalcocite or copper glance and the oxidized ores have been derived.

* *Trans.*, xvii., 479.

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THE DESULPHURIZATION OF PYRITIFEROUS IRON-ORES.

BY STERLING G. VALENTINE, PH.D., LEBANON, PA.

(Colorado Meeting, June, 1889.)

UNTIL within late years, the preparation of sulphurous ores for the blast-furnace has received comparatively little attention. After the first improvement made on the old style of heap-roasting and roasting between walls, there has followed an almost general adoption of the Gjers kiln or some modification thereof, and the majority of iron-masters have seemed perfectly satisfied with so much progress. However, the Westman kiln was developed in Sweden, and other gas-roasters for sulphurous ores have been since constructed. They are in use on the continent of Europe and in America to a limited extent.

The following experiments were made in order to furnish additional information as to the proper conditions for the elimination of sulphur from ores where it is present as sulphide of iron, FeS_2. I desire here to acknowledge the kindness of Dr. Franklin Menges, of the laboratory of Pennsylvania College, at Gettysburg, Pennsylvania, for lending his valuable aid in carrying out a part of these experiments and analyses.

In the first series of these experiments, pure, roughly-pulverized pyrites crystals were used, containing 53.42 per cent. of sulphur. They were made with a view to determine the effect on pyrites, of heat *alone*, in various degrees of intensity and duration, air being altogether excluded. The heats at a low temperature were made in glass combustion-tubes, sealed at the end, where the pyrites to be tested were placed, and with only a small opening at the opposite end. Although not allowing circulation, this arrangement still permitted a part of the evolved sulphur to escape. The heats at high temperatures were made in Hessian and graphite crucibles, tightly luted, every precaution being taken to exclude air. The temperatures attained are given approximately, and are as nearly correct as it was possible to estimate them, using Pouillet's scale as a guide.

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The following table will show the effect of heating pyrites, FeS_2, air being excluded:

Approximate Temperature in Deg. Fahr.	Duration of Heat.	Sulphur in Residue. Per cent.	Loss.	Per cent. of Total Sulphur Expelled.
Original Pyrites.....	.....	53.425	.....	.....
1250	45 min.	45.500	7.925	14.830
"	5 hrs.	42.591	10.834	20.270
1400	1 "	45.140	8.285	17.379
"	1 "	43.570	9.855	18.446
"	6 "	41.850	11.575	21.665
1600	30 min.	37.933	15.492	28.990
1800	1 hr.	39.389	14.036	26.272
2600	30 min.	29.519	23.906	44.720
"	45 "	31.519	21.906	41.003
"	1¼ hrs.	29.960	23.465	43.917

The residues from these heats had lost their metallic luster and were generally grayish in color, soft and friable. The samples heated at a white heat were, of course, completely fused. The figures show that, except in the case of a high heat, comparatively a small part of the sulphur was expelled by heat alone without access of air. Contrary to the idea generally prevalent, it is shown by these results that, while a part of the sulphur volatilizes readily, it is by no means easy to get rid of half of it if air is excluded.

The next experiments were undertaken to ascertain the effect of heat on FeS_2 when air is given access freely:

Approximate Temperature in Deg. Fahr.	Duration of Heat.	Sulphur in Residue. Per cent.	Loss.	Per cent. of Total Sulphur Expelled.
Original Pyrites.....	.....	53.425	.....	.....
1250	1 hr.	4.271	49.154	92.050
"	2¾ hrs.	.700	52.725	98.680
1600	20 min.	.78	52.645	98.540
"	45 "	.08	53.345	99.850
1800	20 min.	.133	53.292	99.751
"	1 hr.	.645	52.780	98.790
2200	15 min.	3.231	50.194	93.952
"	20 "	5.916	47.509	88.926
"	35 "	1.558	51.867	97.102
"	2 hrs.	1.181	52.244	97.780

It will be noted that a larger amount of sulphur remains in the residues when higher temperatures have been applied. I account for this by the fact that, the high temperature being quickly reached,

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a sudden fusion of some of the small particles of pyrites enclosed a portion of the sulphur in such a way as to prevent the action of air upon it.

A high temperature is, therefore, not needed for the complete decomposition of FeS_2 when air has abundant access. We say "complete" decomposition, for, although sulphur still is present in small amounts, yet the expulsion of 98 to 99 per cent. of it is, to all intents and purposes, "complete." Even at a temperature as low as 1250° F., 98.68 per cent. of the sulphur present was expelled when air had free access. At slightly higher heat, 99.85 per cent. was driven off. These results do not bear out the very prevalent idea that, after having got rid of half the sulphur of pyrites, a much higher temperature is required to expel the remainder.

In this connection Ledebur, in his *Metallurgy*, says:*

"Pyrite and marcasite are decomposed even at a low temperature, while about half the sulphur is volatilized (if air is present, burning to SO_2; if not, passing off as vaporized sulphur) and a compound remains whose constitution corresponds nearly to pyrrhotite (Fe_8S_9). By simple heating, without air, this lower sulphide of iron cannot be further decomposed. If, however, atmospheric oxygen has access to the glowing compound of sulphur and iron a further part of the sulphur burns, volatilizing as SO_2; the remainder forms next, with the iron, sulphate of iron, which is decomposed, even at a red heat. A part of the oxygen of the SO_3 is used for the higher oxidation of the iron, basic sulphates of iron oxide are formed, and SO_2 vaporized; on still further raising the temperature, the basic iron salt is also decomposed, SO_3 driven off, and oxide of iron remains behind."

In general, the above-described experiments agree with this statement of Ledebur. But it must be remembered that in saying that "half the sulphur is volatilized" at a low heat, he evidently means the one atom of sulphur in FeS_2. This, however, only reduces the sulphur from 53.33 per cent. in FeS_2 to 39.66 per cent. in Fe_8S_9. My results have shown that this is the actual result, although with a white heat, without air, I have succeeded in reducing the sulphur to a somewhat lower point.

From the two tables given it appears that, for proper desulphurization of pyrites, air is absolutely necessary, and of greater importance than an excessively high temperature.

Experiments made by Dr. Menges as to the amount of air needed for proper desulphurization of FeS_2, gave the following results:

* *Handbuch der Eisenhüttenkunde*, p. 187.

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Working with similar quantities of pyrite in each case, in a current of 900 cubic centimeters of air per minute, the sulphur volatilized without any subsequent condensation in the tube, and on gradually reducing this quantity to 450 c.c., and then to 300 c.c., no condensation occurred. Starting with a current of 400 c.c. per minute, the sulphur volatilized, condensed again further on in the tube, but was subsequently oxidized in the same current without raising the heat. These results show that with ample air-current all sulphur volatilized will readily burn to SO_2, and that in a deficient air-current any sulphur condensed will be subsequently removed.

In view of the fact that many furnace-men are under the impression that pyritiferous ores can be largely desulphurized in the upper part of the blast-furnace, the following experiments were made: Samples of pyrite were heated in a current of gas taken from the "down-take" flue of a furnace. The gas had the following composition by weight:

	Per cent.
Carbonic oxide,	31.17
Carbonic acid,	10.06
Nitrogen,	58.72

The results were as follows:

Approximate Temperature in Deg. Fahr.	Duration of Heat.	Per cent. Sulphur in Residue.	Loss.	Per cent. of Total Sulphur Expelled.
Original Pyrites		53.42		
1250	10 min.	41.14	12.28	22.98
"	1 hr.	33.93	19.49	36.68
1800	25 min.	33.88	19.54	36.58

These figures show that the desulphurization of pyrites in an atmosphere of furnace-gas is, at best, only partial, the results being comparable to those obtained when it is heated without access of air. In view of the already ascertained necessity for oxidizing atmospheric surroundings, it could but be expected that in an atmosphere devoid of free oxygen only partial desulphurization could be effected. The furnace-man who hopes to use the top part of his furnace as an ore-roaster has little success to look for.

When sulphate of iron occurs or is formed in the processes of roasting, the question arises as to its behavior on heating: whether or not the presence or absence of air has any influence. It was, of

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course, known that sulphate of iron on heating decomposes to ferric oxide and sulphur trioxide,* but having seen no statement as to the effect of the presence or absence of air, I made the following experiments:

Two heats, in presence of air for twenty and thirty minutes, respectively, at about 1600° F., gave residues showing 0.031 and 0.030 per cent. of sulphur, 99.87 per cent. of the original sulphur having been expelled.

Three heats, with the air excluded, at somewhat higher temperatures than the above, for thirty, thirty-five and sixty minutes, gave residues showing, respectively, 0.04, 0.04, 0.002 per cent. of sulphur; 99.81 to 99.9 per cent. having been expelled.

Sulphate of iron is decomposed by heat alone, the presence of air or its absence having no influence. From this it may be inferred that sulphate of iron is harmless in the blast-furnace, its sulphur being expelled by heat in the form of SO_3 and passing out with the furnace gases. The harmlessness of sulphate of iron is, indeed, known to most furnace-men, although all are not familiar with the reasons for it.

In view of this behavior of sulphate of iron, we inquire next, how far the manner of heating pyrites affects its presence in the residue. Does the presence or absence of air affect the percentage of sulphur present in the residue in the form of sulphate, which we have found to be harmless? To throw light on this point, residues from some of the heats were analyzed to find how far the sulphur contents was in the form of sulphate of iron.

1. *Pyrites Heated Without Access of Air.*

Temp. ° F Approx.	Duration of Heat.	Total Sulphur in Residue.	Sulphur as Sulphide.	Sulphur as Sulphate.	Per Cent. of Total Remaining Sulphur as Sulphate.
1250	5 hrs.	42.591	42.399	0.192	0.45
1800	1 hr.	39.389	39.247	0.142	0.39
2600	30 min.	29.519	29.485	0.034	0.10

2. *Pyrites Heated in Presence of Air.*

Temp. ° F Approx.	Duration of Heat.	Per Cent. Total Sulphur in Residue.	Sulphur as Sulphide.	Sulphur as Sulphate.	Per Cent. of Total Remaining Sulphur as Sulphate.
1250	2¾ hrs.	0.70	0.231	.469	67.00
1800	1 hr.	0.645	0.158	.487	75.50
2200	2 hrs.	1.181	0.687	.494	41.83

* Roscoe and Schorlemmer, *Treatise on Chemistry*, vol. ii., Pt. 2, p. 101.

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There is a very marked difference in the results. It might be expected that free access of atmospheric oxygen must be a condition for the conversion of sulphide into sulphate. The results show clearly the correctness of that view. Without access of air, practically all the sulphur remaining is in the form of sulphide, but when air has access, a large part, sometimes the larger part, is in the form of sulphate, a fact which, in view of the harmlessness of the latter in the blast-furnace, emphasizes the necessity for abundant air in the roasting of pyritiferous ores. Not only is *more* sulphur removed in this case, but, even of what remains, the greater part is harmless.

In this connection, it was of interest to note the effect of heating in an atmosphere of furnace-gas on the amount of sulphur left in the residue in the form of sulphate. Analyses of the residues from those heats made in a current of furnace-gas, given above, showed as follows:

Temp. ° F.	Duration.	Per Cent. Total Sulphur in Residue.	Sulphur as Sulphide.	Sulphur as Sulphate.	Per Cent. Total Sulphur as Sulphate.
1250	10 min.	41.148	41.089	0.059	0.143
1250	1 hour.	33.882	33.800	0.082	0.212
1800	25 min.	33.936	33.806	0.130	0.382

These results show that this manner of decomposing pyrites is not conducive to the formation of sulphate in the residue, and that the results here, also, are comparable to those got by heating without access of air.

It must be borne in mind that all the above experiments were made on sulphide of iron alone. The question arises whether the conclusions reached on pure pyrites will hold good for a pyritic ore. Of course, sulphide of iron can be expected to act in the same way under the same circumstances under the action of heat. Whether or not an ore containing it will permit desulphurization on the basis of the above-ascertained behavior of FeS_2 depends entirely on the character of the ore itself. If its structure is sufficiently porous to allow air to gain a ready access to the interior of the ore-pieces, it can be expected to desulphurize at a low heat. If it is a close, dense ore, this characteristic is equivalent to exclusion of air from the greater part of its sulphur contents. Dense ores are not difficult to desulphurize because they cannot be heated properly, but because air cannot penetrate them. Therefore, because pyrites can be de-

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composed at a low heat in presence of abundant air, it does not follow that all ores can be desulphurized under corresponding conditions. While it may be said that atmospheric oxygen is a *sine qua non* for proper roasting, no general rule can be laid down concerning the proper temperature to be used.

In order to test the correctness of these views, the following experiments were made: Small pieces of Cornwall ore, about the size of a walnut, containing, as nearly as could be judged, the same amounts of sulphur, were heated at a low temperature under free access of air. The original raw ore showed 2.664 per cent. of sulphur. The following results were obtained:

Temp. ° F.	Duration.	Per cent. Sulphur in Residue.	Loss.	Per cent. Total Sulphur Expelled.
1200	2 hrs.	.346	2.318	87.01
1200	4 "	.178	2.486	93.31
1500	1 hr.	.099	2.565	96.28

From these results it would seem that a comparatively open-grained and porous ore can be readily desulphurized at a comparatively low temperature when air has access freely.

The necessity of an oxidizing roasting so that the sulphur remaining shall, as far as possible, consist of sulphate, is proved by an analysis of the same ore:

Temp. ° F.	Duration.	Per Cent. Total Sulphur in Residue.	Sulphur as Sulphide.	Sulphur as Sulphate.	Per Cent. Total Sulphur as Sulphate.
1200	2 hrs.	.346	.235	.111	32.18
1200	4 "	.178	.088	.090	51.12
1500	1 hr.	.099	.027	.072	72.72

The importance of having the remaining sulphur as far as possible in the form of the harmless sulphate, and of an oxidizing roasting to secure this end, can be readily seen.

Experiments were also attempted with a very dense Jersey magnetite, but it was found impossible, with the stock of it on hand, to select small pieces comparable as to their sulphur contents, and the results were worthless.

One thing, however, was clearly shown, namely, that the density of this ore so completely excluded air from the interior of the ore-pieces, as to make futile any attempt to desulphurize it at a low temperature. It required the use of a high heat.

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The action of the heat itself on the ore probably varies. It may either cause a cracking of the ore-piece and allow the entrance of air in that way, or it may act, as pointed out by Akerman, in enabling the sulphur to take up oxygen from the ore itself, for its oxidation. It is certain that the degree of heat required for desulphurization varies with different ores; but an abundant supply of air should always be present.

It is certainly possible, however, to use too high a heat. It may be questioned whether a pyritiferous ore should ever be heated fully to its sintering point. A fused ore is harder to reduce in the furnace, and heating it to that point appears to be often exceedingly prejudicial to proper roasting.

To test this point—how far fusion or clinkering of ore affects the amount of sulphur that can be expelled, and how far its influence goes to affect the amount of the sulphur remaining in the clinker in the form of sulphate, the following experiments were made: Small pieces of Cornwall ore, similar to those used before, were heated beyond their sintering point, and as quickly as possible, to intensify the effect. For comparison, the former results, obtained on heating at a low temperature are added:

No. of Heat.	Temp. °F.	Duration.	Per Cent. Total Sulphur in Residue.	Sulphur as Sulphide.	Sulphur as Sulphate.	Per Cent. Total Sulphur as Sulphate.
1	1200	2 hrs.	0.346	0.235	0.111	32.18
2	1200	4 "	0.178	0.088	0.090	51.12
3	1500	1 hr.	0.099	0.027	0.072	72.72
4	2400	45 min.	2.125	.....	.....	.....
5	2400	" "	2.422	2.337	0.085	3.50

Recalling the fact that the raw ore contained 2.664 per cent. of sulphur, the effect of the sudden fusion can be readily noted in the differences between the results of the first three heats and of the last two.

The following are analyses of some clinkers taken from roasters working on Cornwall ore, showing the same effect:

Per Cent. Total Sulphur.	Sulphur as Sulphide.	Sulphur as Sulphate.	Per Cent. Total Sulphur as Sulphate.
1.397	1.224	.173	12.380
1.380	1.245	.135	9.782
1.873	1.777	.096	5.125

These results indicate plainly that the fusion of ores in roasting

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may prove prejudicial to good results. When it once takes place it is scarcely possible for desulphurization to proceed further. The viscid coating of fused iron oxide, silica, and lime, or whatever may, in various cases, go to make up the impervious exterior, is a barrier both to ingress of air and egress of sulphur in any form. If the heat in any roasting process must be carried to so high a point, it should be gradually raised, and sintering should take place only after continued heating at a lower temperature in abundant air. I have frequently found pure sulphur condensed in the cavities of cold clinkers, where it had been vaporized from pyrites, but was unable to escape. Clinkering ore quickly is equivalent to roasting without air, and even more, for the sulphur that heat alone would vaporize is imprisoned in the mass. Furthermore, it prevents any great amount of the sulphur left from being in the form of sulphate.

Ledebur states that desulphurization will be the more complete the smaller the pieces of ore, the more freely air has access, and the higher the temperature on roasting.* It is easy to understand that small pieces mean greater surface exposure and less requirement for penetration of heat and air. We have seen that air is absolutely needed for anything like a complete desulphurization. As to temperature, our results do not bear out the asserted necessity of a high heat for thorough roasting in all cases.

The practical application of all these conditions in the roasting of ores is a matter of no little difficulty. A proper roaster must be one in which the heat is under control, the ingress of air ample, and the egress of the products of combustion prompt. A gas-roaster seems to approach most nearly to these conditions. The Gjers kiln in any of its forms seems to be a very deficient apparatus for desulphurization, because it has none of these qualifications. Its effectiveness is about on a par with roasting without air, for little or no air can get at the ore while it is being heated in this style of kiln. Being fired with solid fuel, it is, as a rule, under little control as to temperature, and the greater number of them are chronically "clinkered."

The working of such a kiln can be seen from a few analyses at hand. At the Musconetcong Iron Works, in a modified form of this kiln, the results as given by Mr. I. P. Pardee (*Trans.*, xv., 680), were as follows:

	Per cent. Sulphur.
Raw ore,	0.883
"	0.68
Roasted ore,	0.39
"	0.99

* *Handbuch der Eisenhüttenkunde*, p. 188.

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Cornwall ore roasted in Gjers kilns gives the following results:

	I.	II.	III.	IV.
Per cent. sulphur, . .	.80	1.41	1.05	1.12

As this ore runs from 2½ to 3 per cent. of sulphur, the analyses show only imperfect desulphurization, and are comparable to roasting out of contact with air. As a rule, not more than half the sulphur is removed.

The following conclusions may be drawn from this investigation, in regard to pyritic ores:

1. Heat alone, without access of air, can remove, at best, only one-half of the sulphur present.

2. Atmospheric oxygen is absolutely necessary for a proper desulphurization.

3. Even at a low heat, ore is properly desulphurized if air can gain access freely to the FeS_2 in it.

4. Sulphate of iron can be decomposed by heat equally well with or without air.

5. In order that the residuum of sulphur in roasted ores may consist, so far as possible, of sulphates, the roasting must be done under free access of air.

6. Fusion or sintering of ore is likely to prevent any further desulphurization.

7. Sintering does not allow much of the remaining sulphur to be in the form of sulphate.

8. Fusion, hence, should never occur in roasting except after continued heating in air at a lower temperature.

9. Ores cannot be properly desulphurized in the upper part of the blast-furnace.

10. An efficient roaster must allow easy control of heat, abundant access of air to the hot ore, and rapid removal of the products of combustion.

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NOTES ON THE MANUFACTURE OF OPEN-HEARTH BRIDGE-STEEL.

BY. N. W. SHED, PHŒNIXVILLE, PA.

(Colorado Meeting, June, 1889.)

THE specifications for bridge-steel are usually not severe. The main difficulty is to secure a material which will roll easily, without showing signs of cracking on the corners of the blooms or on the flanges of the beams.

In many mills sulphur has been deemed the most harmful constituent in this respect. It has been thought hazardous to roll steel containing over 0.07 per cent. of sulphur. Yet, in making steel for the Phœnix Iron Company, I have produced from stock running as high as 0.13 per cent. of sulphur a material which rolled very satisfactorily. It is reasonable to infer that the injurious effect of sulphur can be neutralized, to a great extent, by proper treatment in the open-hearth furnace; and the following notes from my usual practice here may be of interest to those obliged to use high-sulphur stock.

The steel made at Phœnixville in the 20-ton furnaces has about the following composition:

	Per cent.
Carbon,	0.15 to 0.20
Manganese,	0.50 to 0.60
Sulphur,	0.11 to 0.13
Phosphorus,	0.04 to 0.06
Silicon,	0.01 to 0.02

The best running heats were made with a small amount of pig-iron. In using large proportions of pig the quantity of ore is also increased. This not only cuts the bottom, but also lengthens the time required to bring the charge down. Up to 50 per cent. of pig may be used, but 25 per cent. is much better. A 40,000-pound charge, containing 10,000 pounds of pig and 30,000 pounds of Bessemer butts or beam-scrap, works very well.

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This is melted as quickly as possible. When every particle of the charge is thoroughly melted and the bath is hot, ore is added. If the ore is added before the charge is melted, the cold ore chills the partly-melted metal, and much time and heat are needed to make the bath perfectly fluid again.

In a charge containing 25 per cent. of pig, 1000 pounds of hematite ore are usually required to bring the carbon to 0.16 per cent. This amount is weighed out, and "feeds" of 200 pounds each are put in as fast as possible, the temperature of the bath being kept up.

With the bath in good condition, all the ore may be put in within one hour. During the addition of ore there should be a lively action throughout the metal; and before all the ore has been put in an energetic "boil" should be visible all over the bath. The heat on the metal is now gradually increased, and the lively action is kept up for nearly an hour. With the decrease in carbon the intensity of the action diminishes and the slag becomes thicker and more viscous. At the end of one hour after the last addition of ore the bath becomes very quiet, the bubbles of gas formed are few, and break slowly through the slag. The metal, if exposed to view, presents a clear, silvery appearance. A test taken at this time usually breaks with a bright, coarsely-crystalline fracture, showing considerable toughness. The carbon is near the 0.16 per cent. point. If it be a few hundredths higher the slight action in the bath will soon bring it to the desired point without further addition of ore. In case it be necessary, however, to add ore at this stage, sufficient time should be allowed, after the ore has entirely disappeared, for the bath to become tranquil before pouring. Steel treated in the furnace in this way should remain perfectly quiet in the moulds when poured.

The ferro-manganese, in small pieces, is shoveled into the ladle while the stream is running out of the furnace. This method of adding ferro-manganese is most economical. The old way of throwing it into the furnace took twice the amount of ferro-manganese to give the same percentage of manganese in the finished metal. In making low steel I have noticed a loss of two-thirds of the manganese when added to the bath before pouring. By adding ferro-manganese in the ladle the loss is practically nothing.

This steel seems to work better when poured cool; but a large number of hot heats have rolled very well. The temperature of the ingot when bloomed down is a good red heat, noticeably colder than is the usual practice.

The steel is top-poured, making ingots 15 by 18 inches in size, and weighing 4200 pounds each.

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The following physical tests were taken at random from a large number of heats. It will be noticed that the increase in the percentage of carbon is not always followed by an increase in tensile strength:

Physical Tests of Open-Hearth Steel.

(*Test-pieces* 8 *in. long,* ¾ *in. in diameter.*)

No.	Tensile strength. Per sq. in.	Reduction of area. Per cent.	Elongation. Per cent.	Carbon. Per cent.
1	57,100	65.3	25	0.15
2	64,100	62.6	26¾	0.18
3	61,200	61	31¼	0.18
4	66,700	53.8	24½	0.21
5	66,700	49.3	23¾	0.19
6	60,000	55.1	26¼	0.15
7	60,800	58.7	24¼	0.13
8	67,500	46.3	24¼	0.16
9	69,260	45.4	22½	0.22
10	60,320	54.1	30	0.17

THE INFLUENCE OF SILICON ON THE DETERMINATION OF PHOSPHORUS IN IRON.

BY THOMAS M. DROWN, PROFESSOR OF ANALYTICAL CHEMISTRY IN THE MASSACHUSETTS INSTITUTE OF TECHNOLOGY, BOSTON.

(Colorado Meeting, June, 1889.)

THE process for determining phosphorus in iron now in most general use in the laboratories of iron and steel works, is, I think, the one proposed by Mr. Emmerton.* In this process the solution of the iron for precipitation by ammonium molybdate is effected by treating the iron with nitric acid, evaporating to hard dryness in a dish, taking up with hydrochloric acid and driving off the greater portion of the hydrochloric by nitric acid. This procedure accomplishes, first, the destruction of all the carbonaceous matters; and, second, it renders all the silica insoluble.

Experience has proved the former—the complete oxidation of the carbon—to be absolutely essential; but the explanation usually given, namely, that unless the carbon compounds are destroyed they will have a solvent action on the yellow precipitate, I am inclined to

* *Trans.*, xv., 93.

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doubt. It seems to me more probable, as suggested by Mr. P. W. Shimer, that the conditions which are favorable for the oxidation of the carbon compounds ensure the complete oxidation of the phosphorus to ortho-phosphoric acid, and that the reason of the imperfect precipitation of the phosphorus, when the solution of the iron in nitric acid is directly precipitated by molybdate solution, is that the phosphorus is not completely oxidized.

There have been many means proposed for destroying the carbon compounds—chromic acid, potassium chlorate, potassium permanganate, hydrogen peroxide and others—which accomplish the purpose without the necessity of evaporating the solution to dryness. The advantage of the latter method, however, has been supposed to be that in this way the silica was at the same time rendered insoluble.

The statement of Klein* that, in the determination of phosphoric acid in basic slag, it is not necessary to separate the silica, since the ammonium silico-molybdate is soluble in 3 per cent. nitric acid, led me to try the effect of precipitating the phosphorus in a solution of pig-iron without previously evaporating to dryness. The results agreed perfectly with others obtained in the usual way after the removal of the silica; and further investigation showed that the silicon, at least in the form in which it exists when pig-iron is dissolved in nitric acid, is not precipitated by the molybdate solution.

There is a very marked difference in the effect of nitric acid of different strengths on pig-iron, as regards the condition of the silicon. When the acid is strong (1.2 to 1.4 sp. gr.) much of the silicon becomes gelatinous and prevents filtration, while with more dilute acid the silicon goes almost entirely into solution.

In a former communication to the Institute† I gave the effect of different strengths of hydrochloric acid on the silicon in pig-iron, showing that the strongest acid renders nearly all the silicon insoluble, and that weak acid dissolves it almost entirely. Generally speaking, the same is true, also, of nitric acid, as the following determinations show. It is interesting to note in the following figures that a nitric acid of medium strength dissolves more silicon than acid either stronger or weaker.

The pig-iron used in the following tests was a No. 1 foundry-iron, containing 2.42 per cent. of silicon.

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* *Chem. Zeitung*, 10, 721. † *Trans.*, viii., 514.

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Effect of Nitric Acid of Different Strengths on the Silicon in Pig-Iron.

Acid.	Silicon, Per cent.		Acid.	Silicon, Per cent.	
Sp. Gr.	In Graphitic Residue.	In Solution.	Sp. Gr.	In Graphitic Residue.	In Solution.
1.4	2.28* 2.08*	0.14 0.34	1.102	0.12 0.12 0.12 0.12 0.13	....
1.2	0.75* 0.44*	1.67 1.98			
1.16	0.07 0.07	....	1.082	0.23 0.23 0.23	....
1.135	0.07 0.06 0.06 0.06	....	1.070	0.20 0.24 0.27 0.31	2.12 ..... 2.14 2.12
1.116	0.09 0.11	....			

The solutions obtained with the use of acid of 1.4 and 1.2 sp. gr. could not be filtered in concentrated condition, owing to the gelatinous form of the silica. They were, therefore, largely diluted and an aliquot portion of the solution was obtained by filtration. In this the silicon was determined.

The solutions obtained with acid of 1.116 sp. gr., and still more dilute, can be filtered without difficulty.

The separation of gelatinous silica is not wholly a matter of strength of acid, neither is it wholly a matter of the percentage of silicon in the pig-iron. It has often been noted that different pig-irons, under identical treatment with nitric acid, will filter at different rates; and sometimes the iron with least silicon proves most refractory in filtering. This suggests that silicon may exist in different conditions in pig-iron, a subject which cannot be further discussed in this paper.

Another pig-iron was tested in this series of experiments—a gray-forge iron, containing 0.63 per cent. of silicon—with the following results:

* By difference.

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Silicon in Graphitic Residue.

	Per cent.
Acid, sp. gr. 1.2,	0.19
	0.20
Acid, sp. gr. 1.135,	0.03
	0.03
	0.05
	0.06

From the foregoing it will be seen that the strength of nitric acid most favorable for the solution of pig-iron is about 1.135 sp. gr., and not 1.2, which is generally used.* In the solution thus obtained there is no tendency of the silica to separate; not until the solution has been concentrated by boiling to small bulk does the silica begin to gelatinize.

When molybdate solution is added to a solution of pig-iron obtained in this way, the phosphorus, as might be expected, is but partially precipitated, because this treatment with nitric acid is insufficient to oxidize all the carbonaceous matters; or, as I prefer to believe, insufficient to oxidize all the phosphorus to ortho-phosphoric acid. To effect this object any of the oxidizing agents already mentioned may be used. Potassium permanganate, as proposed by Shimer,† seems to me the most convenient; but for the solution of the separated manganese peroxide I use, instead of hydrochloric acid a small quantity of some organic substance, such as tartaric, citric, or oxalic acid or sugar. The manganese oxide disappears almost immediately on the addition of the organic matter. After this treatment, which requires only a minute or two, the molybdate solution will give a complete precipitation of the phosphorus.

The following determinations were made in order to discover whether there is any tendency for the silicon in solution to be carried down with the yellow precipitate, either mechanically or as silico-molybdate. The pig-iron used was that already mentioned, containing 2.42 per cent. of silicon.

Silicon determined	I. Per ct.	II. Per ct.	III. Per ct.	IV. Per ct.
In residue from nitric acid solution,	0.040	0.040	0.070	0.100
In yellow precipitate,	0.030	0.050	0.012	0.012
In washings of the yellow precipitate with 2 per cent. nitric acid,	0.050	0.030	0.007	0.004
In filtrate from the yellow precipitate,	2 250	2.210	2.340	2.280
Total,	2.370	2.330	2.429	2.396

* Acid of 1.135 sp. gr. is conveniently made by diluting acid of 1.2 sp. gr. with half its bulk of water, or by diluting acid of 1.4 sp. gr. with water in the proportion of 3 to 7.

† *Trans.*, xvii., 100.

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There seems to be, then, no evidence of the formation of any precipitate of ammonium silico-molybdate.

If the separation of the silica is omitted, the determination of phosphorus in pig-iron may be carried out in very much less time than is ordinarily required. In the laboratory of the Massachusetts Institute of Technology we have adopted the following procedure: About 1.5 grammes of the pig-iron (or more if very low in phosphorus) is treated in a covered beaker with 60 c.c. of nitric acid of 1.135 sp. gr. (If preferred, 20 c.c. of water may be first added, and then 40 c.c. of nitric acid, sp. gr. 1.2. In this variation the action is at first a little less violent.) The beaker is heated on an iron plate until action ceases; three minutes is usually sufficient time if the borings are reasonably fine. The solution is then filtered from the graphite into a 500 c.c. flask, and to the boiling filtrate is added 15 c.c. of permanganate solution of 5 grammes to the liter.* Boiling is continued until the pink color has disappeared, and a few very small fragments of tartaric acid are then added, causing the precipitated manganese peroxide to go almost immediately into solution. The amount of tartaric acid need not exceed 0.1 gramme, although the addition of an excessive quantity—as much as 1 gramme—does not influence the results perceptibly. To the clear solution 10 c.c. of strong ammonia (sp. gr. 0.90) are added to partially neutralize the free nitric acid. This amount leaves the solution still amber-colored; it should not be so far neutralized as to become wine-colored. When the solution has cooled down to 90° C., 80 c.c. of molybdate solution† is added, and the flask is closed with a rubber stopper, wrapped in a towel and shaken vigorously for five minutes, as recommended by Emmerton. From this point on the directions of Emmerton, in his paper already mentioned, are followed, namely, reduction by zinc and titration by permanganate. If fine, powdered zinc is used it may be entirely dissolved in about ten minutes, but filtration from the undissolved zinc seems generally to be preferred.

The following results were obtained in this way on the sample of pig-iron containing 2.42 per cent. of silicon, in comparison with the older method of dissolving the iron in nitric acid, evaporating to

* The strength of this solution is not a matter of importance: I have found the above amount sufficient. Shimer uses 10 c.c. of a solution of 20 grammes to the liter.

† Made by dissolving 100 grammes of molybdic acid in 400 c.c. ammonia, sp. gr. 0.96, and adding 1000 c.c. nitric acid, sp. gr. 1.2.

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dryness to separate silica, and precipitating by molybdate solution and magnesia mixture.

No. 1 Pig-Iron Containing 2.42 Per cent. of Silicon.
Phosphorus Determinations.

By the Old Method.	By the New Method.	By the New Method (Continued.)	
Per cent.	Per cent.	Per cent.	Per cent.
0.284	0.272	0.297	0.286
	0.279	0.296	0.294
	0.290	0.289	0.294
	0.280	0.291	0.288
	0.291	0.291	0.292
	0.292	0.278	0.288
	0.289	0.294	
	0.292	0.293	

The rate of oxidation of the reduced molybdenum sesquioxide, on exposure to the air, is very slow, and is not ordinarily taken into account. Mr. J. W. Loveland found that it amounted, in this series of experiments, to 0.001 per cent. of phosphorus for every ten minutes standing before titration with permanganate. All the foregoing results have been corrected accordingly.

If rapidity of determination is of the first importance, a few minutes may be saved by omitting the filtering off of the graphite. Many trials show that this omission does not affect the accuracy of the results, although the removal of a foreign insoluble substance from the liquid enables one to carry out the process with greater satisfaction.

The following determinations of the phosphorus in the sample of gray-forge iron containing 0.63 per cent. of silicon, were made without filtering off the graphite, by Mr. Greeley, a student at the Massachusetts Institute of Technology.

Gray-Forge Iron Containing 0.63 Per cent. of Silicon.
Phosphorus Determinations.

By the Old Method.	By the New Method, without filtering from graphite.	
Per cent.	Per cent.	Per cent.
0.632	0.620	0.627
0.635	0.625	0.629
	0.631	0.645
	0.640	0.635
	0.633	

Two determinations made in the solution of the same pig-iron, after filtering from the graphite, gave 0.636 and 0.631. Further results obtained by Mr. Greeley are given below:

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Bessemer Iron Containing 1.18 *Per cent. of Silicon.*
Phosphorus Determinations.

By the Old Method. Per cent.	By the New Method. Per cent.
0.084	0.083
0.087	0.085
	0.086
	0.086
	0.085

No. 1 *Pig-Iron Containing* 2.29 *Per cent. of Silicon.*
Phosphorus Determinations.

By the Old Method. Per cent.	By the New Method. Per cent.
0.334	0.324
0.330	0.327
	0.334
	0.324
	0.322

The process for steel is, of course, the same as for pig-iron, except that the question of filtration from graphite does not arise.

In a sample of Bessemer steel the following percentages of phosphorus were obtained.

By the Old Method. Per cent.	By the New Method. Per cent.
0.061	0.064
	0.060
	0.061

By one familiar with the details of the process, a determination of phosphorus in pig-iron can be easily made in less than an hour.

I am indebted to Mr. James W. Loveland, Assistant in Chemical Analysis in the Institute of Technology, for valuable aid in this investigation; the credit for the analytical work, except as otherwise mentioned, is due entirely to him.

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AN IMPROVED FRENCH POCKET-COMPASS.

BY R. A. BERGIER, BUTTE CITY, MONTANA.

(Colorado Meeting, June, 1889.)

THE *Transactions* of the Institute contain nothing, as yet, on the subject of pocket-compasses; and in the belief that American miners, explorers, geologists, and engineers will gladly welcome any information about new forms of apparatus, simple in construction and adapted to rapid use, I offer a short description of an improved French combination pocket-compass, which, if once introduced into use, would seem likely to supersede similar instruments of older pattern. The instrument in my possession I owe to the kindness of M. G. de La Bouglise, of Paris.

The frame of the compass is a double-bottomed box, $2\frac{1}{2}$ inches square and $\frac{9}{16}$ of an inch high, made of white metal, and with its surfaces very carefully planed. The sides of the box make true right-angles with each other. Two sides are parallel with, and two are normal to, the north and south line of the principal graduation. The bottom is accurately planed, so that the instrument will lie true on a plane surface. A glass, $2\frac{3}{8}$ inches in diameter, is set in the top plate.

The graduation for the needle is engraved on the upper face of a ring, connected with the flat disk, in which the center-pin is inserted. Between this disk and the bottom of the box there is a hollow space a quarter of an inch deep.

The needle is $1\frac{7}{8}$ inches in length, and can be balanced by means of a small sliding piece of metal. The graduation extends from 0° to 360°, reading from right to left. The stopping of the needle is effected by a screw, *A*, on the side of the box near the handle, *H* (Fig. 1). There is also a clinometer attachment, the graduation for which is engraved on the upper face of the silver-plated disk.

The compass has two sights. The north sight, *B*, is a simple pin, a little more than an inch high, which, when not in use, is kept folded down, and is partially protected by the exterior sliding-plate, *C*. When raised for use, the sight is stopped in a vertical position by a projecting pin attached to the side of the box. The south sight, *D*, is connected with the handle, *H*, and the sight and handle

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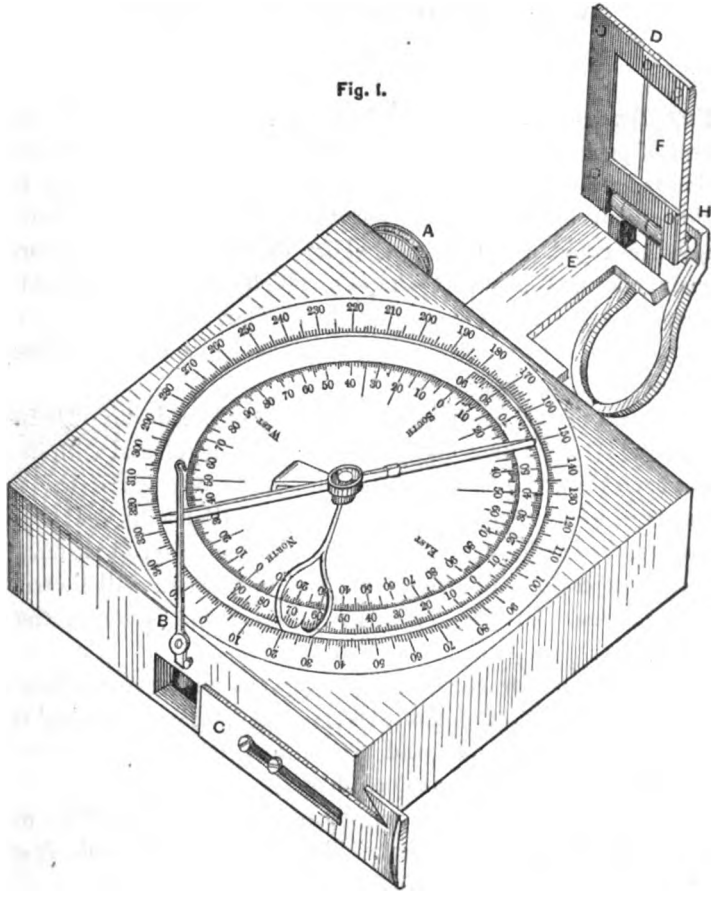
together are hinged to the extremity of a plate, *E*, an inch and a half long by half an inch wide, which can be slid through a slot in the side of the box, so as to rest on the bottom plate in the hollow space referred to above. From one side of the plate, *E*, a rectan-

Fig. I.



gular piece of metal has been cut for use in leveling, as will be explained further on. When the compass is not in use, the sight is folded down so as to lie flat on the sliding-plate, and the two are pushed into the box, so that nothing but the handle is visible on the outside. The sight itself, *F*, is a rectangular piece of glass, 1¼ inches high, so set in its frame that it has two reflecting surfaces, one on each face of the sight; the slit cut in the center lies in the vertical plane passing through the north and south line of the compass.

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The distance between the two sights may be made longer than the side of the compass by sliding out, to its full length, the movable plate, *E*.

The handle, *H*, is flat and pear-shaped, and is movable about its hinge, so as to be capable of taking various positions, the extremes being when reversed and lying flat under the bottom of the box, and when standing perpendicular to the face of the compass.

The opening on the north side of the box, shown in Fig. 1, but not lettered, leads to a chamber in the hollow space beneath the disk. This chamber serves as a place of storage for a small ball-and-socket joint, one end of which can be screwed into a hole in the compass bottom directly under the center-pin, while the other end is provided with a screw by which the instrument can be attached to any convenient wooden support in either a horizontal or vertical position, according to the observation to be taken.

The compass is carried in a nice wooden case lined with velvet.

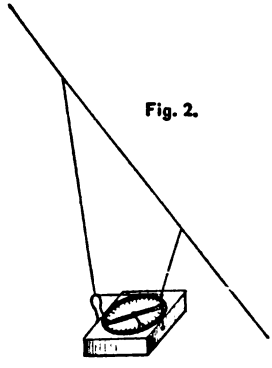

Fig. 2.

On the inside of the cover there is a mirror, in the middle of which a line has been scratched and colored red (Fig. 3). When the case is closed, this line covers the north and south line of the compass. The whole apparatus weighs eleven ounces.

When the compass is to be used for rough work only, it need not be taken out of its case; the cover is simply opened, and the eye is so placed as to see on the red line of the mirror the reflection of the object sighted to, additional accuracy being secured by raising up the north sight, *B*. Closer results are obtained when the compass is taken from its case and placed on some flat surface, and both sights are used; but in order to secure the maximum of accuracy obtainable, the socket-joint must be used. By this means the compass can be fairly well leveled, and bearings can be taken with more pre-

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cision. Any kind of a rest, even a walking-cane, can be used to screw the socket to.

When used as a leveling instrument, the sliding-plate, *E*, must be drawn out, and the south sight must lie flat upon it, the compass being held by the handle and allowed to hang freely from the hand (Fig. 4). In this position the sight will be in a vertical plane, and consequently the object seen through the vane, and the image of the eye in the mirror, will be in the same horizontal plane. Leveling can also be done with both faces of the sight, a small screw serving to adjust the sight in a true vertical plane.

The square form of the instrument facilitates its use as a clinom-

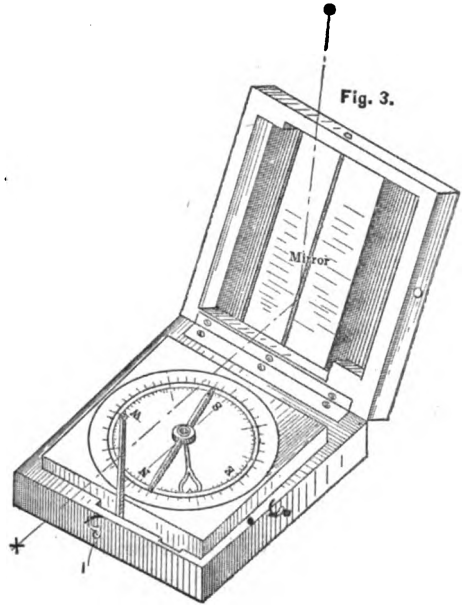

Fig. 3.

eter, the dip or vertical angle being taken by resting the proper side on the surface or line whose inclination is to be measured. A vertical angle can be more accurately measured by using the socket-joint, screwed horizontally upon any appropriate stationary support, situated above or below the object sighted to.

On the back of the compass a table is engraved, giving the percentage of rise corresponding to given angles.

The instrument, as described, is made by M. Dutrou, 18 Rue Dauphine, Paris, and a patent has, I think, been applied for in the United States. I cannot give its exact cost, but I think it is about

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twenty dollars at the place of manufacture. In its present form it makes an excellent pocket-theodolite, deserving of broad acceptance, but a few additions or changes might, perhaps, be introduced for use in this country, such as:

(1.) A lighter and more strongly magnetized needle, in order to lessen the momentum of the oscillations, and admit of shortening the time of observation.

(2.) The introduction of a graduation showing bearings directly, to take the place of, or to be added to, the 360-degree graduation.

(3.) By adding a hook to the north sight, and notching the handle, the instrument could be suspended from a line, and used in nearly

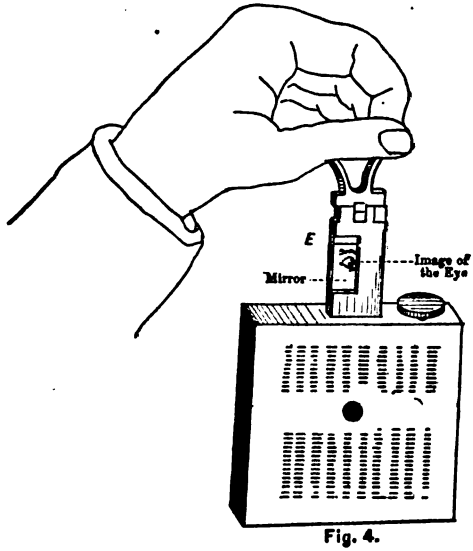

Fig. 4.

the same way as the German mining compass for taking horizontal or inclined courses in mines (Fig. 2).

The compass box being square, the plotting of observations on paper can be done at once, the lines being drawn with one side of the box as a rule, or, if the side is too short, by laying a longer rule against it. With the aid of this compass topographical data may be determined, if not with great accuracy, at least with considerable ease and rapidity. The ascertaining of the height of a summit and its horizontal distance from the place of observation, becomes a simple graphical operation after the necessary plotting of a previously measured base has been properly carried out.

A good, but not indispensable improvement would be to make the graduation circle movable around its axis, in order to set off the variation of the needle.

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ALUMINUM IN CAST-IRON.

BY W. J. KEEP, DETROIT, MICH.

(Analytic Determinations in Charge of C. F. Mabery.)*

(Colorado Meeting, June, 1889.)

THIS paper is prepared to substantiate the statements made by us in a paper upon the same subject last year,† and to present the chemical analyses of the test-bars belonging to the series of casts reported in that paper.

We have proved our assertions by other tests, made in a different manner, and with different materials. We shall show, separately, what influence was exerted by the aluminum and the silicon, which we introduced by the Cowles metal.

We propose, also, to render our former paper more intelligible by additional explanations.

I. GENERAL DESCRIPTION OF TESTS AND RECORDS.

We have found that our graphic representations were, in some cases, not perfectly understood. In the tables by which we illustrate all our tests, the records appearing in the vertical columns between the rulings are the results of the tests of iron having the percentages and characteristics noted at the top of the column. The record is made by a short horizontal line in such vertical columns. It is located in horizontal spaces, which divide the table into units of measure, to indicate the value of the record. For example, in the upper part of Table VII. is recorded the transverse strength of the series of tests with white iron and aluminum, which were reported in our former paper. The numbers at the extreme top are the test-numbers. The fractional numbers below are the percentages

* The analyses for this paper were made under my direction by Mr. L. D. Vorce, a student in the Case School of Applied Science.—C. F. M.

† "The Influence of Aluminum Upon Cast-Iron." Read before the American Association for the Advancement of Science, at the August meeting, 1888, by W. J. Keep, of Detroit, Mich. Reprinted in full in: *Engineering and Mining Journal*, August 18, 1888, with editorial; *Iron Age*, August 22, 1888; *Metal Worker*, August 22, 1888; *Engineering*, London, Eng., August 31st, with editorial; *Iron*, London, Eng., September 31, 1888.

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of aluminum, which we aimed to introduce into the castings. The decimals below these fractional numbers show the aluminum actually found by analysis. In the first left-hand vertical column we state what the record represents. In the next column the figures indicate the unit of measure represented by the horizontal line located opposite each number. In this transverse record the numbers indicate pounds avoirdupois. The heavy record-lines represent the transverse strength of our bars. They are ½ inch square and 1 foot long, supported at both ends, a weight being gradually applied at the center until fracture takes place. After these short horizontal record-lines are made in each vertical column, their ends are joined, making the continuous full black record-line.

This table contains another record of transverse strength, represented by the heavy broken line. This is the breaking weight of the same sized bars after being subjected to a series of blows, each greater than the preceding. Instead of representing this in inch-pounds, as a separate record, we have given an arbitrary value to an inch-pound in pounds avoirdupois, and have reduced each inch-pound record to that unit. We assumed that the equivalent of the blow given by a 25-pound weight swinging on a 6-foot radius and falling 2⅛ inches is 288 pounds, and then calculated the value of each ⅛-inch variation of the fall. This is simply to permit a comparison, in the same table, of tests by weight and by impact. We do not claim that the force of a blow can at present be stated in pounds.

The object in representing graphically in one table the records of a large number of tests is not to construct a curve of the whole record, as in such case the vertical columns should be of a width represented by the percentages of the element under consideration in each test; but it is to permit the eye to observe at a glance the general effect of the element in the whole series, as well as in each member. As only general conclusions can safely be drawn, this is a fair representation.

In Table VII. we have put the records of the strength of tests 363 and 471 in figures, in order to limit the length of the table.

We do not reduce the records to pounds per square inch, because any one can make this calculation to reduce our results to any length of bar and cross-section. Our records are the averages of the actual breaking-weights of two or three of our bars.

In Table VII. we show the deflection of the same bars. The record-line represents the total bend at the center of our test-bar at the time of fracture. The full line is for weight, and the broken

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line is the deflection for impact. The deflection in decimals of an inch is measured from the position of the center of the bar before testing. As our autographic machines record the exact motion of the bar multiplied five times, we measure the deflection with a 20-scale and record as hundredths. This divides any inaccuracy of measurement by five. The record in the table is the average of the same bars from which we made the strength-record.

The next record in Table VII. is that of shrinkage in decimals of an inch per linear foot of the test-bar, the mould for which was $12\frac{1}{8}$ inches long. The full line represents the shrinkages of the $\frac{1}{2}$-inch square bar, and the broken line that of a thin bar measuring $\frac{1}{10}$ by 1 by $12\frac{1}{8}$ inches, cast with the square bar. The record of each of these bars is an average of either four or six bars, according to the size of the cast.

The record of chill which we have next shown is the depth in inches of the chilled end of the $\frac{1}{2}$-inch square bars. The bars were cast in ordinary sand moulds, but their ends were run against chills exactly $12\frac{1}{8}$ inches apart to allow for the shrinkage. By splitting the bar the depth of chill can be measured.

The fluidity is measured by the actual length in inches of a strip cast in a mould $\frac{6}{100}$ of an inch thick, 1 inch wide, and 12 inches long. The iron is run from one end, the mold lying horizontally, and rarely runs the full length. This shows the fluidity of that cast, which, however, may be influenced by temperature, aside from the effect of variations in the composition of the iron. We endeavor, however, with ordinary irons, to have the temperature as nearly uniform as possible for each test.

The last record in Table VII. is that of the relative hardness. This is measured on a Turner machine by the number of grams placed upon a cutting diamond to cause it to produce a scratch upon the polished surface of the fracture at the middle of a 1-inch square bar.

We offer here a few words of explanation regarding the tables in our first paper. The series of tests, 363 to 367 and 368 to 375, were made by first filling a crucible and then adding to this mass, after each cast, as much new iron and ferro-aluminum as was needed to reach the desired percentage of aluminum in the next cast, and have enough metal for the last cast. We supposed, when we prepared our first paper, that to determine the influence due to the aluminum we must make a comparison-series under exactly the same conditions and with the same weights of material, but without adding aluminum.

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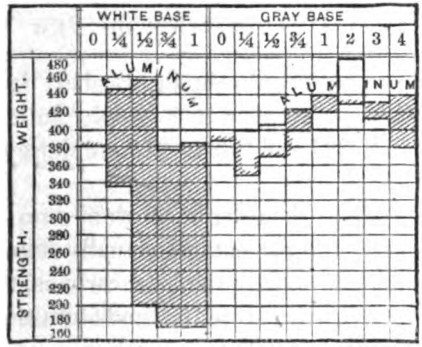
We then represented the tests of each aluminum-series and the corresponding comparison-series on one chart. For the purpose of illustration, Fig. 3 of our first paper is introduced, that figure representing the same record of transverse weights as in Tables VII. and VIII. of this paper. We placed the word aluminum over the upper line of weights to show that they were the records of the aluminum-series. The lower line is that of the comparison-series. As we then held that the influence due to the aluminum was represented by the difference between these records, we shaded the space between the records to show that the result was in favor of aluminum, or left the space white to indicate that the result was un-

FIG. 3.



favorable. In each series we began with a melt of the base-iron, either F^L M or white. The test 363, which began the series of white iron with aluminum, gave very low results, because the casting was full of blow-holes from low silicon (.08), while the cast 376, beginning the comparison-series, though of the same lot of white iron, contained more silicon (.25), and therefore gave a much more solid casting with correspondingly high results. In our first paper we adopted this highest record as that for white iron. In Table VII. we give the result of the test 363 of the white cast, which belonged to the aluminum-series, and compare subsequent tests with it.

We have often called the comparison-series a remelt-series, because a portion of the metal was remelted at each heat. But since new white iron was added at each heat, the series is not truly a remelt-series, but more nearly corresponds to ordinary foundry practice in making light castings, where new iron is melted each day with the sprues and bad castings of the previous day. In Fig. 2 of our first paper the horizontal line opposite .50 should not have been heavier

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than the rest, and the numbers which are misplaced should have been opposite each horizontal line. In this case the aluminum record-line is the lowest, and the space between records is shaded because it is desirable to decrease chill.

In Fig. 3 of our first paper the whole chart records strength to resist weight, and not, as appears from our lettering, the lower part strength and the upper part weight. The same remark will apply to Fig. 4, that being a record of impacts. In our present paper we have reproduced both the record of the white base and of the gray base, with aluminum, in Tables VII. and VIII., but have not thought best to reproduce the record of the physical tests of the remelt-series.

II. General Description of New Tests.

When referring to a vertical column, we shall indicate such vertical column by the test-number at its top. In all tables in this paper the tests 363 to 367 and 368 to 375 are those of the first paper reproduced.

Table IV. shows both the calculated aluminum and silicon contents of each test, and the percentages actually found by analysis.

Table V. exhibits graphically the total carbons, the graphitic and the combined carbons in the white iron and aluminum-series, as well as the same carbons found in the comparison- and remelt-series of our first paper. Table VI. gives the same carbons for the F^{L}M and aluminum-series, and its remelt comparison-series.

In both these tables the heavy full line in the diagrams at the top of the table gives the total carbons in the aluminum-series, while the full light line shows the total carbons of the remelt-series.

In the next lower diagram the full heavy line indicates the graphite in each cast with aluminum, while the heavy broken line exhibits the graphite in the remelts.

The combined carbons of the aluminum-series are shown by the full light line, and those of the remelts by the light broken line in the lower part of the tables. Referring to Tables II. and VII., test 367 represents the last cast of the white aluminum-series, in which aluminum contents of 1 per cent. was aimed at, but which actually contained 1¼ per cent. Before receiving the results of these analyses we made the remainder of the tests of Table VII. to prove the preceding results. As test 367 was made up of white cast-iron and Cowles ferro-aluminum, we first determined whether a cast made of

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white iron and Cowles metal, proportioned to give 1 per cent. of aluminum, would give the same results in a new and independent heat, and whether our previous method of a comparison-series was correct. The result of this experiment in shown in test 473, and is such as to be almost equal to a chemical analysis, proving all that we stated in our first paper, and that the comparison-series was unnecessary. The next question is to ascertain to what extent this result is due to 1 per cent. of aluminum or to the $\frac{5}{8}$ of 1 per cent. of silicon which was introduced with the aluminum by the use of the Cowles metal. To do this we found a pig-iron containing nearly the same (3.64 per cent.) silicon as the Cowles metal (3.86 per cent.) and about the same carbons. To the same white base as before we added enough of this pig to bring the silicon to about $\frac{1}{2}$ per cent., this being the amount supposed to have been in test 367. This cast is shown in 471, and to make comparison convenient we have preceded this test by 363.

We also made another test, 469, with the same white cast-iron and commercially pure aluminum to show the effect of the 1 per cent. of aluminum. The aluminum was kindly donated for our experiments by Charles F. White & Son, 96 Maiden Lane, New York, who are agents for the Aluminum und Magnesium Fabrik, of Hemelingen, bei Bremen, Germany.

The next question was whether the combination found in the Cowles ferro-aluminum was superior to aluminum and silicon and carbon brought together from other sources. To determine this, cast 472 was made, using the same white cast-iron, introducing silicon and carbon by the pig-iron used in cast 471 and adding the aluminum furnished by Messrs. White & Son. This cast should be almost identical in composition with 473; and the record shows it to be so. We made two more casts, one 468 with $\frac{3}{4}$ per cent. aluminum, using the same metals as in cast 469; the other, 470, a remelt of the material left after casts 468 and 469.

Our object in presenting the records of these tests graphically is to allow the observer to perceive at a glance the tendency of any element throughout the whole series, and to avoid the danger of being misled by the exceptions which occur in all tests.

That it may be conveniently located for purposes of reference, we introduce in tabular form the analytical determination of the series of tests presented in our first paper.

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Table I.

CALCULATED				ACTUAL PERCENTAGE FOUND IN TEST BARS									
NO. OF TEST	ALUMINUM	SILICON	SILICON FOR ACTUAL AL.	ALUMINUM	SILICON	TOTAL CARBON	GRAPHITE	COMB'D CARBON	NO. OF TEST	SILICON	TOTAL CARBON	GRAPHITE	COMB'D CARBON
WHITE BASE IRON AND ALUMINUM									WHITE REMELTS				
PIG	.00	.186	.18	.00	.186	2.98	.95	2.03	PIG	.186	2.98	.95	2.03
363	.00	.18	.18	.00	.08	2.98	1.27	1.71	376	.25	3.13	1.62	1.51
364	.25	.26	.27	.25			1.37		377		2.97	2.14	.83
365	.50	.35	.36	.54		2.84	1.35	1.49	378		3.12	1.53	1.59
366	.75	.43	.47	.89		3.10	1.81	1.29	379		2.75	1.83	.92
367	1.00	.52	.62	1.28	.62	2.99	1.71	1.19	380	.12		1.57	
									381	.12	2.52	1.44	1.08
FLM GRAY IRON AND ALUMINUM									FLM REMELTS				
PIG	.00	1.25	1.25	.00	1.249	3.55	3.22	.33	PIG	1.249	3.55	3.22	.33
368	.00	1.25	1.25	.00	1.245	3.41	2.26	1.15	382	1.160	3.97	2.81	1.16
369	.25	1.33	1.28	.10		4.09	2.72	1.37	383		2.61	2.37	.24
370	.50	1.41	1.29	.14		3.55	2.66	.89	384		3.86	2.98	.88
371	.75	1.50	1.37	.32		3.53	2.78	.75	385		3.57	2.60	.97
372	1.00	1.58	1.50	.75		3.45	2.56	.89	386		3.65	2.77	.88
373	2.00	1.92	1.75	1.50		3.58	2.50	1.08	387		3.37	2.32	1.05
374	3.00	2.26	1.99	2.23		3.34	2.76	.58	388		3.72	2.49	1.23
375	4.00	2.70	2.54	3.84	2.280	3.10	2.44	.66	389	.978	3.53	1.82	1.71

Eng'd by American Bank Note Co. N.Y.

III. Influence of Aluminum Upon the Grain of Cast-Iron.

We present this portion of our subject at this stage because the object of "Keep's Test" is to present to the eye both the physical and chemical peculiarities of the iron. The appearance of the fracture is a strong indication of the character of the metal, and one of the advantages in using a bar half an inch square is that it, more than any other size, shows by the change of grain the effect of varying constitution. A smaller bar chills so quickly that it does not give an element sufficient time to exert its influence, while a larger bar holds its heat so long that any tendency towards a white grain is more or less overcome. The grain of the fractures of our tests is shown in Tables II. and III., and represents each test with aluminum

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recorded in Tables VII. and VIII. We began our white series with the cast of a white iron 363, which gave us castings full of blow holes. The area of the bar is so lessened by these holes as to give very low results.

Test 364 is the same white pig-iron with enough of the Cowles ferro-aluminum added to bring the aluminum up to .25 per cent. The result is a more even grain, with absolutely no blow holes. With a glass magnifying 20 diameters, it seems as though minute black grains could be seen scattered throughout the otherwise white surface, and the latter itself seems to have a darker shade. The change in carbons which the analysis shows to have taken place might give a darker grain, and yet the graphite is in such minute grains and in such small quantity that it does not decrease shrinkage. In this cast the silicon also increased, and by so doing added to the hardness of the casting.

This test, 364, is the most important member of the series, for it illustrates the effect of the use of aluminum in foundry practice. It shows how, with a small percentage of aluminum and a little silicon, we can produce extremely hard and perfectly sound castings of white iron, and at the same time obtain the maximum strength.

In test 471 we obtained by an addition of silicon alone a casting nearly as sound but not so strong, especially in its resistance to shock. With silicon at about .20 and aluminum .25 per cent., we have in test 364 a perfectly homeogeneous white grain.

In another test reported in our first paper we produced perfectly sound castings with $\frac{1}{10}$ per cent. aluminum, its resistance to weight being 545 pounds and to impact 254 pounds.

By bringing the aluminum up to $\frac{1}{2}$ of 1 per cent. in test 365, enough carbon has been driven from the combined to the graphitic form to make the fracture gray at the middle of the bar. In Tables II. and III. the character of the central cross-fracture is shown by the square figures on the left side of the tables. The percentage of aluminum and the test-number are shown at the left of each cross-section, and the same bar is shown flat-ways at the right of its cross-section. All of the square bars are cast with both ends against a chill, the chilled end of the bar being placed in these illustrations next to the cross-section. The latter is the fracture at the middle of the bar 6 inches from each chilled end. In the longitudinal representation of the bars, only enough of the length is shown to indicate the changes of grain. The grain of the chilled end of the bar is shown by splitting off the side of the bar.

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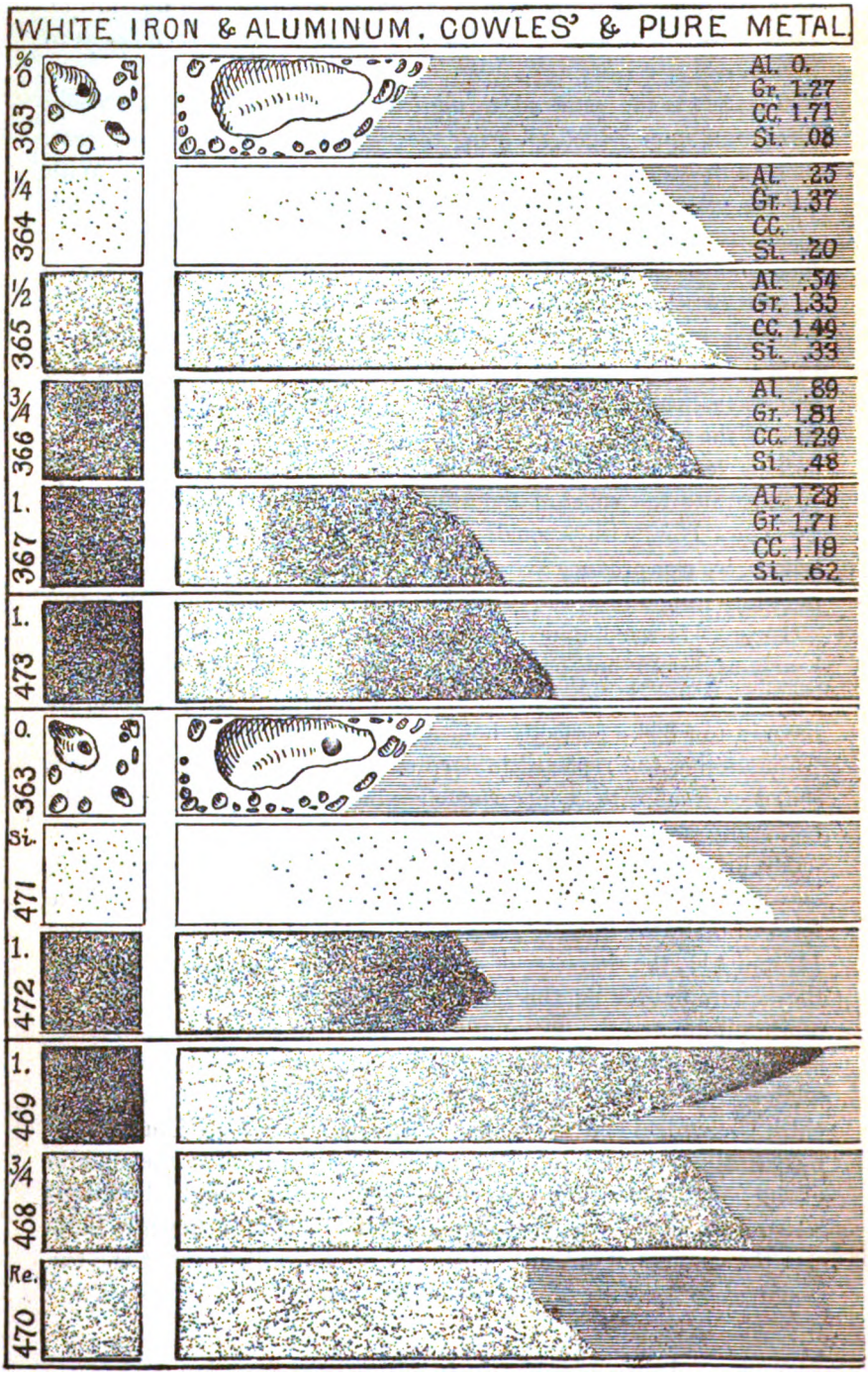
Table II.



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In test 366, with ⅛ of 1 per cent. of aluminum, the cross-fracture is decidedly gray, and the long fracture shows that the gray of the cross-fracture grows less and less until hardly any gray remains. The measure of the white portion of the long fracture shows the depth of chill.

In tests 363, 364 and 365 all or nearly all of the bar was white, so that no measure of the depth of chill could be made. In 366 enough of the bar is gray to allow the depth of chill to be measured.

In test 367, with 1¼ per cent. of aluminum, the fracture is quite dark, and the chill has diminished to a depth of ½ an inch.

This completes the original white series.

The next test, 473, was made to prove 367. The result is the same as 367, except that the chill is deeper. If this test were placed between 367 and 366, it would exactly fill the place of a 1 per cent. metal.

The next three tests were made for the purpose of segregating the effect of the aluminum from that of the silicon.

In test 471 we introduced into the white iron the same proportion of silicon and carbon as we did in 473. The white, 363, was full of blow holes, while 471 was solid, and the grain seemed exactly like 364.

We next made test 472 to be an exact duplicate of 473, but it was made up of 471, with enough commercial aluminum to bring the percentage up to 1 per cent. The result is a lighter-colored cross-fracture than 473, but the chill is not so deep. Considering the fact that in 472 the mixture could not be nearly as complete as in 473, we see the most conclusive proof that 1 per cent. of aluminum and ½ per cent. of silicon in such a white iron as we have used can be expected at all times to produce substantially the same results, and will turn a white porous iron into a solid gray casting.

The question now naturally presents itself whether the aluminum would produce a gray casting without the help of silicon.

Tests 469 and 468 were made to determine this. In test 469 we added to a crucible of melted white iron 1 per cent. of aluminum. This gave at the point of fracture a darker gray than test 472 or even 473, but the chill-fracture shows that the gray diminished rapidly towards the chill.

This may have been due to the fact that the silicon in cast 472 was acting throughout the whole time of melting, while in both 469 and 468 the aluminum was introduced just before the iron was

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poured. In test 473 both the silicon and aluminum were introduced cold with the iron, and both therefore acted during the whole time of melting and cooling.

Test 468 was intended to furnish a 1 per cent. cast exactly like 469, but analysis yielded only .77 per cent. aluminum. The fracture is gray, and this cast, with no addition of silicon, would very nearly in every respect fill the .75 per cent. place between 365 and 366.

We have thus proved that aluminum alone will turn combined carbon into graphite.

We believed that if we gave more time for the aluminum of tests 468 and 469 to act upon the carbon, the iron would be grayer. We therefore remelted together the sprues of these two casts in test 470. The result did not sustain this view, the casting being in grain almost exactly like 365.

Table III. indicates the action of higher percentages of silicon and aluminum than is shown by the tests in Table II. The iron of test 368 contains no aluminum, but carries 1¼ per cent. of silicon. The cross-fracture is a beautiful gray, as the silicon was introduced in the blast-furnace under the most favorable conditions for producing gray iron. In the casting the chill is $\frac{6}{10}$ of an inch deep. The first addition of the Cowles metal took out more than half the chill, and slightly darkened the fracture.

The next three additions, 370, 371 and 372, reduce the chill to nearly nothing, but do not materially darken the fracture. In 373 the chill nearly disappears, and in the last two it is all gone.

A noticeable effect on the grain is the dark color immediately back of the chill, the sudden cooling preventing the grains from assuming any size or grouping together.

IV. Analytical Determination of Aluminum and Silicon.

Having proved the correctness of the assertions which we made in our first paper by proof-tests, we will now present the determinations of aluminum and silicon in the test-bars of our first series. We give these results and other data graphically in Table IV., and in tabular form in Table I. To explain the lack of uniformity in these determinations, we wish to review the method of making these tests.

We would have been glad to have put the whole iron used in the series into the crucible, and after melting down the whole to have

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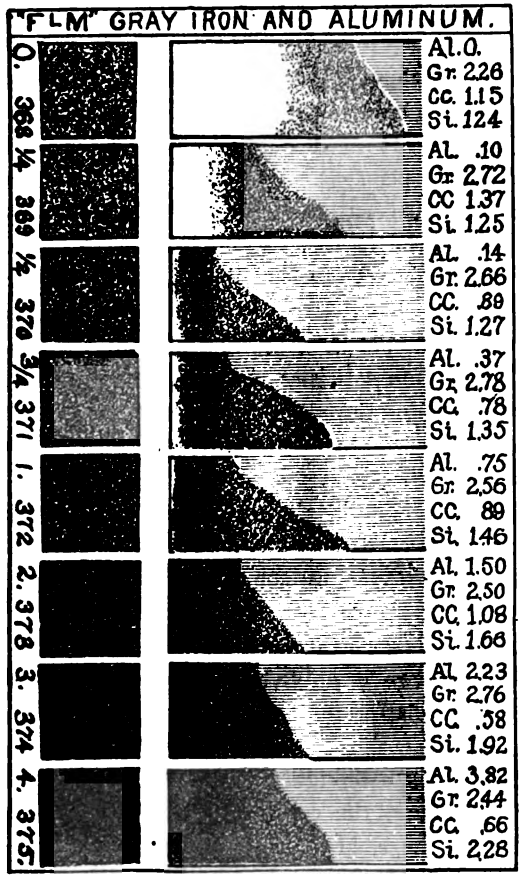
made one cast of the iron alone, and then added the aluminum to make the percentage as desired for the next cast, then re-heated and cast the second test, and so on. But owing to the difficulty in handling so large a quantity of iron at once, and doing accurate work, we put into the crucible from 15 to 20 pounds of the iron, which was used as a base, and after melting made the first cast; then we added a

TABLE III.



new portion of the base iron, and enough of Cowles ferro-aluminum to bring the aluminum in the second cast to ¼ of 1 per cent., and so on. In the selection of irons to be used as the base of experiments, we have taken great care that the metal should be as nearly uniform as possible, but the results show that in the same pigs of white iron, silicon varied from .08 to .25. In F^{L}M, though all pieces appear

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alike, the variation in total carbon shows that it is impossible to produce iron of uniform composition. No two sets of drillings from the same casting are likely to give the same analytical results. We have found that it is impossible to run eight heats, one after the other, and make the composition of the separate melts correspond exactly to the calculated percentage of any element. Therefore no significance is to be attached to *any one record* of a loss or gain from the calculated amount of any element.

Again, the accuracy of the determinations of aluminum in quantities considerably less than 1 per cent. must be regarded as somewhat uncertain, but they have an especial value in showing that this metal remains in the iron in approximately the proportions in which it was introduced. It need not, therefore, be assumed that the amounts less than 1 per cent. are as small as they appear to be in the $F^{L}M$· series. A peculiar feature of this series was that there was very little to skim off before pouring, and there was sufficient metal to run easily the 4 per cent. heat, leaving afterwards in the crucible the amount that it was calculated should remain. Another reason why we think that the aluminum in each cast nearly conforms to the calculated amount is that the ferro-aluminum was added so as to increase the aluminum $\frac{1}{4}$ of 1 per cent. at each heat up to 1 per cent., and afterwards by an amount to increase it 1 per cent. each time.

If there is a deficiency in aluminum in any of the first heats, it would cause the same deficiency in each subsequent heat, but we noticed that the shortage is 60 per cent. of the calculated amount in the $\frac{1}{4}$ per cent. heat, 50 per cent. in the $\frac{1}{2}$ and $\frac{3}{4}$ per cent. casts, and 25 per cent. in the 1, 2 and 3 per cent. heats. In the last, or 4 per cent. cast, the shortage is only .18, or about 5 per cent. of the calculated amount. Instead of augmenting, the shortage grows less at each cast.

In the white series, although the calculations and weighing were made as carefully, and the conditions attending the melting were the same as with the $F^{L}M$ series, yet there was a large amount to be skimmed before each cast.

Although we expected to have been able to carry the series up to 4 per cent., no metal was left in the crucible after the 1 per cent. cast. We supposed at the time that the aluminum was lost, but the analysis proved that it remained. This loss of material at each heat caused the additions of aluminum to be too great. The correspondence between actual and calculated aluminum in the (white)

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¼ and ½ per cent. melts is only apparent, as they are averages of two determinations for each, viz., .32 and .18 for the first, and .62 and .45 for the latter. We have dwelt at length on this matter, to show that only general conclusions can be drawn from any series of casts, because no cast-iron is of the same composition throughout, and because other irregularities of composition are produced in the cooling of a cast.

We also note in Table IV. the percentages of silicon that should have been present at each cast. The silicon in the first and last cast of each series was determined analytically. Then by plotting the curve, shown by the full silicon line, we arrived at the probable percentage in each cast.

V. General Review of Chemical and Physical Tests.

The Relation Between Carbon Contents and Strength.—Taking the $F^{L}M$ series, the pig contained 3.20 per cent. of graphite and only 0.33 per cent. of combined carbon. On remelting (test 368) the loss in silicon is practically nothing, yet the carbons are 2.26 per cent. graphite and 1.15 combined carbon. The remelting temperature was probably lower than that of the blast-furnace, and the size of our bars was less than that of the pig. This gave less time to the graphite in our bars to separate.

As all tests are made at the temperature of our furnace, the carbons of this first heat are those of the original pig cast under the conditions attending our tests. Taking the $F^{L}M$ aluminum-series as a whole, we see that combined carbon decreases rapidly as compared with the first melt. Graphite which stood at 2.26 per cent. at the first heat (368) ran up to 2.78 with ¼ per cent. aluminum (371) and afterwards diminished rapidly with each heat, but not so rapidly as combined carbon decreased. The reason is that the total carbon has decreased very rapidly as the tests progressed. By taking this into account, we see that the proportion of graphite to combined carbon has increased as aluminum increased. This proves that aluminum converts combined carbon into the graphitic form while the metal is still melted, and that much carbon has risen to the surface during each melting and thus been lost. The increasing blackness of the fracture of each cast also shows that as the temperature lowers graphite continues constantly to be formed. Aluminum thus drives carbon out of cast-iron. An examination of carbons of the remelt-series of $F^{L}M$ shows that the effects on the carbon are exactly the opposite to those in the $F^{L}M$ aluminum-series. In the remelt-series

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graphite decreased while combined carbon increased, and total carbon remained substantially the same as in the original pig. The silicon introduced with the aluminum also drove out carbon, but the chief expulsive action was that due to the aluminum.

We stated that if the casting was sound and not too brittle the less graphite the more strength. The carbon results show that this is not only true, but that a throwing out of graphite from the melted iron and consequent decrease in the total carbon adds to strength. Any treatment therefore that will decrease the total carbon will be likely to strengthen the iron.

We have in another paper* shown that while silicon will act in the same manner upon carbon, yet by its own weakening effect it impairs the strength of the iron as it increases in amount more than the decrease of carbon strengthens it. From the results of our experiments we are led to believe that the aluminum of itself does not add to the strength of the cast-iron into which it is introduced.

Aluminum through its influence on carbon seems indirectly to increase the strength of cast-iron. It will be noticed in the tests which we have reported that the strength increases as total carbon decreases, and this is in spite of the cutting up of the mass by the increasing proportion of imprisoned graphite. The white series with the aluminum also corroborates this. Combined carbon decreases very rapidly, and graphite increases as rapidly with aluminum.

These series, with the new tests presented in this paper, and the actual determinations of carbon, prove that aluminum (in these series aided somewhat by silicon) produces these changes in carbon.

We began with a white iron that produced castings full of blow holes, and therefore weak from that cause. The strength of the $\frac{1}{4}$ and $\frac{1}{2}$ per cent. casts was due to the fact that the casting was solid. The iron was low in carbon to begin with, and consequently the iron was not fluid. This prevented the carbon from escaping to the surface, and caused the total carbon to remain practically the same throughout the series. Since, therefore, it did not gain in strength from a decrease of total carbon, and since the proportion of graphite was greater, the iron was necessarily weakened in subsequent heats. Neither of these series should have been compared with the remelt-series, because in the latter the blow holes increased with each melt. The growing weakness was due to this cause. We have proved also by the direct melt 473 in Table VII., as compared with the last melt

* "Silicon in Cast-Iron," *Trans.*, vol. xvii., p. 683.

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Table IV.

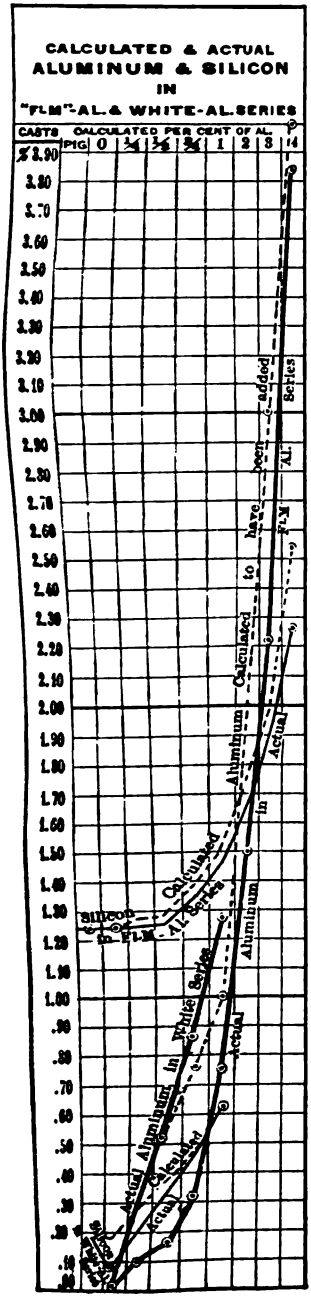


Table V.

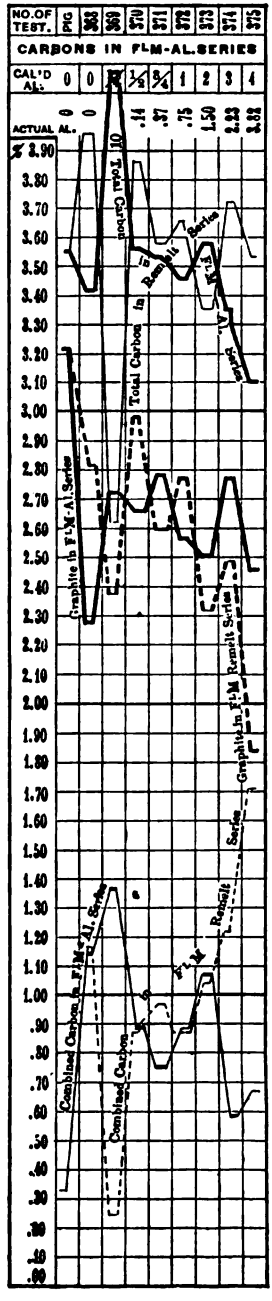
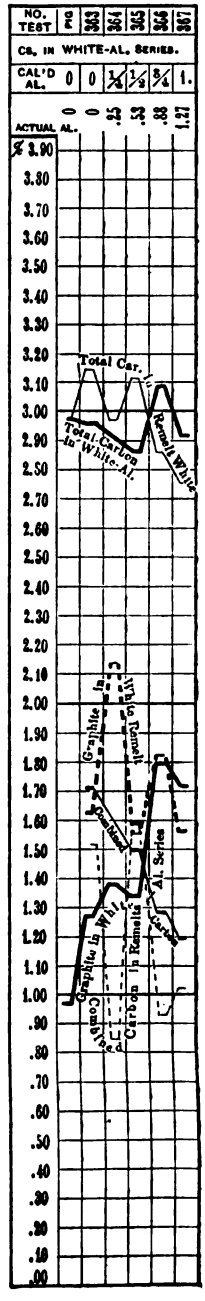


Table VI.



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of the series (367), that each member of the series should be viewed alone, or compared with the first cast without aluminum (363), and with the other members of its own series. We also see from this record that in this series all variation in strength can be traced to the variation in quantity and condition of the carbon. This white series proves better than the F^LM series that aluminum changes combined carbon into graphite, and thus prevents brittleness, and tends to produce sound castings.

Deflection.—This is not dependent upon the total carbon. It varies directly as the graphite, and inversely as the combined carbon.

Chill.—The chill increases in depth as combined carbon increases and as graphite diminishes. The F^LM pig, the analysis of which showed little combined carbon, had a very dark even fracture, and the thinnest edges of the pig were as dark as the heavy portions. The changes produced by remelting (368) illustrate the tendency of the carbon to the combined state. Eight remeltings turned the iron white on account of this natural tendency, and not from a decrease of silicon, for in the first remelting there was no change in silicon. The noticeable thing is that the addition to F^LM of from $\frac{1}{10}$ to $\frac{1}{4}$ of 1 per cent. of aluminum increases the graphite to such an extent, and in such a way, that the iron loses its tendency to chill. The 3 and 4 per cent. melts have absolutely no chill. Silicon will not take out all tendency to chill.

Shrinkage.—This decreases as combined carbon is lowered. We have found that when aluminum is added in small quantities shrinkage increases perceptibly. This is apparent in both of these experimental series. Aluminum of itself, when combined with carbon in cast-iron, probably exerts some influence towards increasing shrinkage, but the carbon soon changes into graphite to such an extent that such tendency to shrink (if it exists) is overcome, and shrinkage is thus indirectly reduced by aluminum.

Relative Hardness.—This was determined by a Turner machine. In the paper on the "Influence of Silicon," to which we have before referred, it was shown that in those series of tests hardness increased with the combined carbon and with the silicon, except during the period when the softening, brought about by the changing of combined carbon into graphite through the influence of silicon, more than counteracted the hardening effect of the silicon itself. Thus far the indication is that aluminum of itself hardens cast-iron. An examination of the shrinkage-record in a cast, when silicon had been increased or decreased, proved that shrinkage varied as the hardness.

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Table VII.

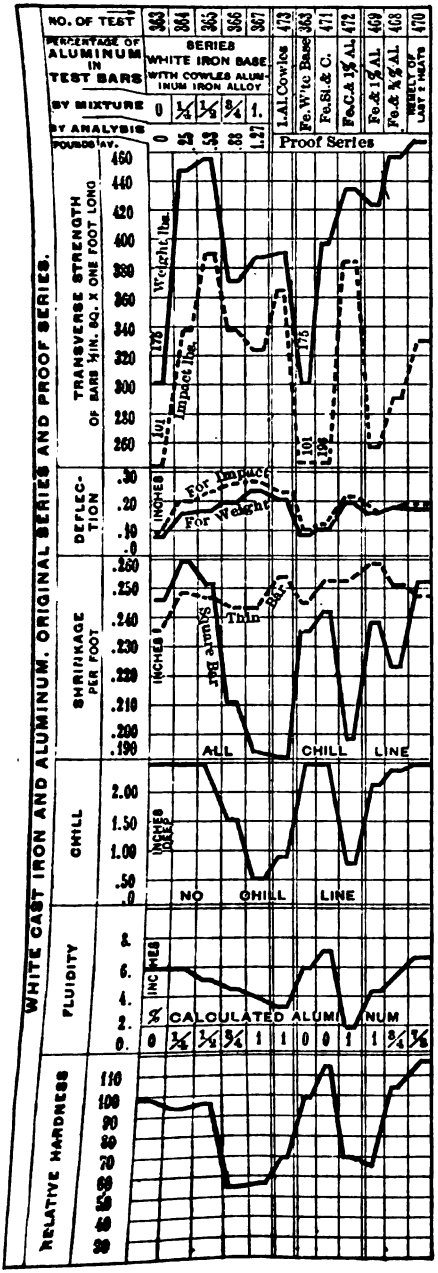
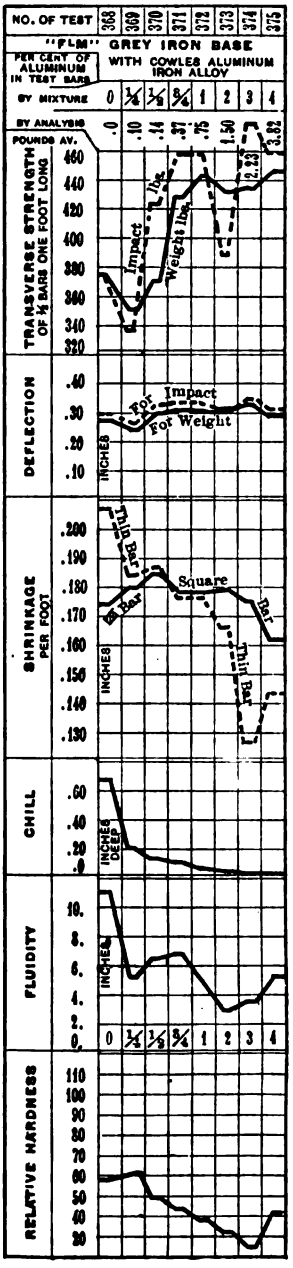


Table VIII.



An examination of the shrinkage-record of the tests recorded in this paper shows that hardness and shrinkage are very similar. The in-

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crease of shrinkage of the thin bar indicates hardness, even though in larger castings slower cooling leads to a separation of sufficient graphite to cause the square bar to have a low shrinkage. An iron that will make thin castings with high shrinkage, and larger castings with a low shrinkage, will be found harder than when the shrinkage of the thin and thick are nearer together. The softest metal, where aluminum is the agent for changing carbon, is that where the thin castings shrink least.

By taking into account the shrinkages of both thin and square castings in each member of the proof-series, it will be observed that hardness and shrinkage seem to be caused by the same elements. The Wellston pig, by which we endeavored to make the carbon and silicon of 471 the same as 473, was a gray iron. The white iron to which it was added had a hardness ranging from 60 to 100, yet the union of these two to bring the silicon up to .54 per cent. does not seem to be sufficient to liberate graphite, and thus makes a casting combining a very high shrinkage with extreme hardness. The same mixture, melted under the same conditions and charged with the same amount of aluminum (pure) as was contained in cast 473, made a casting (472) with low shrinkage and quite soft. The high shrinkage of the thin strip in casts 473 and 472 shows that the combined carbon is not as low as in 367. These tests prove that during the time that aluminum is changing combined carbon into graphite, it softens cast-iron.

Fluidity.—It is not an easy matter to explain from the chart of carbons the decrease of fluidity produced by the introduction of aluminum. An extended use of aluminum as alloyed with cast-iron indicates that the decrease is due to an effect upon the surface of the melted metal. As the metal containing aluminum flows through the mould, it seems to have a skin drawn in front of it. The Cowles 10 per cent. metal, if it contained no aluminum, would, no doubt, be an exceedingly fluid iron, but with its aluminum it cannot be run into castings as small as our thin test-bars. The metal poured on a slab will solidify about ½ an inch thick with the sides round. When cold it has the appearance of being covered with a skin which is drawn into wrinkles. The ordinary commercial aluminum cools in the same way, although it makes a sharp casting. This surface-tension causes a tendency in large drops of the metal to solidify in spherical shapes. Whenever two currents of iron containing aluminum, however small in amount, come together in the mould, they are apt not to unite, but form "cold shuts."

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VI. The Use of Ferro-Aluminum in Foundry Practice.

Practice.—The object of these papers upon the influence of aluminum is not to encourage the introduction of aluminum into cast-iron when it is remelted, but to point out the fact that aluminum, if present in pig-iron, would change combined carbon into graphite, as silicon does, but without weakening effects. With our present experience, the introduction of aluminum into molten cast-iron is attended with serious difficulty.

If ferro-aluminum broken into very small pieces is introduced into the ladle before the iron is caught, it will probably be melted as the first iron strikes it; but, if plunged into the iron after the iron is caught, it is likely to chill a coating of iron around it. In this condition the metal swims. The attempt to melt such chilled pieces in the ladle of iron is useless. Commercially pure aluminum thrown on the surface of cast-iron melts at once, and forms a film on the surface which seems to sink into the iron and form a true alloy. Yet the whole amount thus introduced does not seem to remain in the iron, as it does when successfully introduced by the ferro-aluminum. The surface and the sides, after the iron is poured out of the crucible, seem to be covered with aluminum. Aluminum introduced into the foundry-ladle in the smallest quantities causes the iron to boil rapidly, probably because the particles of graphite which it liberates are light. This ebullition is so rapid that slag which has risen to the surface is often carried under again by the force of the current.

A more unfortunate effect is that the film of oxide and slag covering the surface of the iron in the ladle is made so tender by the aluminum that it does not hold together while the iron is being poured. As a result, patches of scum break away and flow with the iron into the mould, doing more harm than if such impurities had not been cleared from the iron. It might be better if the aluminum could be introduced into the hearth of the cupola before the iron is tapped, so that it could have time to cause a separation of slag, leaving the slag inside the furnace. We have endeavored to overcome these difficulties by introducing the ferro-aluminum with the charges of pig-iron, but this is not a desirable method, if a better one can be found.

The value of our discovery regarding the action of aluminum upon carbon is to show the desirability of the introduction of aluminum into foundry pig-iron during its reduction in the blast-furnace,

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and to call attention to the fact that this is the most desirable element in such irons, if an iron without chilling tendencies is sought.

If introduced into the pig-iron during its reduction in the blast-furnace, as silicon is now introduced, none of the evils referred to would be encountered.

A misconception seems to prevail as to the use of aluminum, even if it were not attended with the practical difficulties referred to. It is not to gray, but to white iron, or iron low in silicon, that aluminum can be added with advantage. Any addition of aluminum to irons already made gray by silicon, and consequently weak, would cause a further separation of graphite, and consequently make the iron grayer and weaker. While silicon and aluminum work together to make iron gray, yet if the silicon be left out and aluminum used instead, the iron will be grayer, stronger and softer than if silicon were the agent. We do not advocate the addition of anything to pig-iron unless it be unavoidable. Purchase pig-iron that will of itself, or when mixed with other brands, make such castings as are desired without any doctoring. If irons can be found that are made gray by aluminum, such irons would be very desirable for the making of thin castings where strength as well as softness is desired.

THE COAL-TRADE AND MINERS' WAGES IN THE UNITED STATES FOR THE YEAR 1888.

BY CHARLES ALBERT ASHBURNER, ASSISTANT IN CHARGE OF COAL STATISTICS, U. S. GEOLOGICAL SURVEY, PITTSBURGH, PA.

(Colorado Meeting, June, 1889.)

THE coal-fields of the United States have been variously classified as to their geographical positions. In 1887 I proposed slight changes to the classification generally used, for more convenient description of our coal-fields in my report on the statistical history of coal in the States during 1886 for the United States Geological Survey. This new classification seems to meet all practical requirements and has been generally adopted. The coal-areas are grouped in two divisions—the anthracite division and the bituminous division.

The anthracite division, in a commercial sense, may be said to include the anthracite districts of Pennsylvania alone, although small amounts of anthracite are mined in Colorado and have been

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mined in Arkansas. In the New England basin the original coal-beds have been metamorphosed into graphite and graphitic coal, which have special uses, although not classified by the coal-trade as anthracite.

The bituminous division includes the following coal-fields: (1) The Triassic field, embracing the coal-beds of the Triassic or New Red Sandstone formation in the Richmond basin in Virginia, and in the coal-basins along the Deep and Dan rivers in North Carolina. (2) The Appalachian field, which extends from the State of New York on the north to the State of Alabama on the south, having a length northeast and southwest of over 900 miles, and a width ranging from 30 to 180 miles. (3) The Northern field, which is confined exclusively to the central part of Michigan. (4) The Central field, embracing the coal-areas in Indiana, Illinois and Western Kentucky. (5) The Western field, including the coal-areas west of the Mississippi river, south of the forty-third parallel of north latitude and east of the Rocky Mountains. (6) The Rocky Mountain field, containing the coal-areas in the States and Territories lying along the Rocky Mountains. (7) The Pacific Coast field, embracing the coal-districts of Washington Territory, Oregon and California.

The table on page 124 contains the approximate areas of these coal-fields, as far as it has been possible to estimate them from the maps of government geological surveys, with the total product of each, in short tons of 2000 pounds (exclusive of colliery consumption), during the calendar years of 1887 and 1888 respectively.

Coal-Product in 1888.

The total product of all kinds of commercial coal in 1888 was 142,037,735 short tons (increase over 1887, 18,022,480 tons), valued at the mines at $204,221,990 (increase, $30,625,994). This may be divided into: (1) Pennsylvania anthracite, 43,922,897 short tons (increase, 4,416,642 short tons), or 39,216,873 long tons, including 38,145,718 long tons shipped by the railroads and canals, and reported by their statistician, Mr. John H. Jones, and 1,071,155 long tons sold to the local trade at the mines, valued at $85,649,649 (increase, $6,284,405). (2) All other coals, including bituminous, brown coal, lignite, small lots of anthracite produced in Colorado and Arkansas, and 4000 tons of graphitic coal mined in Rhode Island, amounting in the aggregate to 98,114,838 short tons (increase, 13,655,833 tons), valued at $118,572,341 (increase, $24,406,589).

The colliery consumption of the individual mines varies from

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The Coal-Fields of the U. S., their Areas and Product in 1887 *and* 1888.

	Area.	Product in 1887.	Product in 1888.
ANTHRACITE.	Sq. Miles.	Short Tons.	Short Tons.
New England (Rhode Island and Massachusetts)	500	6,000	4,000
Pennsylvania	470	39,506,255	43,922,897
Colorado	.....	36,000	44,791
	970	39,548,255	43,971,688
BITUMINOUS. (a)			
Triassic.			
Virginia	180	30,000	33,000
North Carolina	.....	.....	.....
Appalachian.			
Pennsylvania	9,000	30,866,602	33,796,727
Ohio	10,000	10,301,708	10,910,946
Maryland	550	3,278,023	3,479,470
Virginia	185	795,263	1,040,000
West Virginia	16,000	4,836,820	5,498,800
Kentucky	9,000	950,903	1,193,000
Tennessee	5,100	1,900,000	1,967,297
Georgia	200	813,715	180,000
Alabama	8,660	1,950,000	2,900,000
	58,695	55,193,034	60,966,240
Northern.			
Michigan	6,700	71,461	81,407
Central.			
Indiana	6,450	3,217,711	3,140,979
Kentucky	4,000	982,282	1,377,000
Illinois	36,800	10,278,890	14,655,188
	47,250	14,478,883	19,173,167
Western.			
Iowa	18,000	4,473,828	4,952,440
Missouri	26,887	3,209,916	3,909,967
Nebraska	3,000	1,500	1,500
Kansas	17,000	1,596,879	1,850,000
Arkansas	9,043	150,000	276,871
Indian Territory	.....	685,911	761,986
Texas	4,500	75,000	90,000
		10,193,034	11,842,764
Rocky Mountains, etc.			
Dakota	.....	21,470	84,000
Montana	.....	10,202	41,467
Idaho	.....	500	400
Wyoming	.....	1,170,318	1,481,540
Utah	.....	180,021	258,961
Colorado	.....	1,755,735	2,140,686
New Mexico	.....	506,034	626,665
		3,646,280	4,588,719
Pacific Coast.			
Washington	.....	772,612	1,215,750
Oregon	.....	31,696	75,000
California	.....	50,000	95,000
		854,308	1,385,750
Total product sold	.....	124,015,255	142,037,735
Colliery consumption	.....	5,960,302	6,621,667
Total product	.....	129,975,557	148,659,402

(a) Including lignite, brown coal, and scattering lots of anthracite, not included under that head.

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nothing to 8 per cent. of the total output of the mines, being greatest at special Pennsylvania anthracite mines, and lowest at those bituminous mines where the coal-bed lies nearly horizontal, and where no steam-power or ventilating furnaces are used. The averages for the different States vary from 2 to $6\frac{4}{10}$ per cent., the minimum average being in the Pennsylvania bituminous, and the maximum average being in the Pennsylvania anthracite region.

The total output of the mines, including the colliery consumption, was:

	Long tons.	Short tons.	Spot value.
Pennsylvania anthracite, .	41,624,611 or	46,619,564	$89,020,483
All other coals in U. S., .	.....	102,039,838	122,497,341
Totals, . . .		148,659,402	$211,517,824

On March 1, 1889, the United States Geological Survey published a brief preliminary report showing the total product of coal in each State and Territory, with corresponding spot values. These provisional figures, which I made up from incomplete returns and estimates, were as follows: Total product, including colliery consumption, 145,363,744 short tons; spot value, $208,129,806. The final results, made up from complete returns received up to June 15th, show an increase over the preliminary estimates of 3,295,658 short tons and $3,388,018 spot value for the entire product of the United States.

The Coal-Trade During 1888.

The growth in the coal-trade of the United States during the year 1888 was not generally so notable as during the previous year, either in the increased tonnage or the increased value, although the growth in special localities was greater than had ever before been realized. During 1888 there were 13,655,833 short tons more coal shipped from the mines to market than during the previous year, as against 13,523,266 tons increase in 1887 over 1886. The tonnage increase in 1887 and 1888, therefore, was practically the same, yet the per cent. of increase in 1887 was greater than in 1888.

The increase in the spot value of shipments during 1888 was $24,406,589, while in the previous year the increase in spot value was only $18,676,123. When the coal sold to the local trade around the mines and used for mining is taken into account, the increase in the total product during 1888 was 18,683,845 short tons, and the increased value was $28,960,987 as against 17,232,154 short tons and $27,956,661 spot value for the previous year.

The most notable increase anywhere in the United States was in

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Pennsylvania, and especially in Allegheny county, in the bituminous region; the increase in the anthracite region was also very great, showing an excess over 1887 of 4,416,642 short tons, or 3,943,431 long tons; the total product of the anthracite region being sold at an increase of $6,284,405 over the value of the total sales for 1887. This increase in the anthracite regions of Pennsylvania is all the more notable, when it is remembered that one of the most disastrous strikes which has ever taken place in any of the mining regions in America occurred in the anthracite region during the early months of the year.

With the exception of Georgia and Indiana, more coal was produced in each State and Territory during 1888 than during the previous year, and the value of the product was greater during the latter year in every district,* with the exception of Georgia.

The spot value of anthracite coal per ton at the mouth of the mine during 1888 is estimated to be 6 cents lower than during the previous year, while in the bituminous regions of Pennsylvania there was an increase in the value of 5 cents per ton. In most of the other districts the spot value of the coal during 1888 ranged from 3 to 11 cents per ton more than during 1887.

The advantages of anthracite coal for domestic consumption are becoming every year more apparent, and the areas over which the Pennsylvania and Colorado anthracites are distributed are rapidly increasing, while consumption in old areas is growing from the natural growth in population. In no way is this fact better realized than by the increased popularity which the capital stocks of the anthracite-producing companies is attaining among conservative capitalists.

The *Engineering and Mining Journal*,† in speaking of the anthracite stocks at the opening of 1889, says:

"Upon Wall Street the term 'coal stocks' is usually applied to the leading anthracite stocks. During the year under review this group has been very prominent in the speculative arena, and holders of these stocks have more cause for congratulation than the holders of any other group of stocks represented at the New York Stock Exchange. During 1887, the tendency of the prices of these stocks was downward, while in 1888 the tendency was upward, and in some cases higher prices were attained than have been recorded for years past. The improvement in the prices of these stocks was not evenly divided—Reading, Lehigh Valley and Lehigh Coal and Navigation were adversely affected by a long strike at their mines, while the other stocks were favorably affected by the same cause. Owing to this strike,

* No account is taken here of Rhode Island and Idaho.

† Vol. xlvii., p. 39, January 12, 1889.

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Delaware, Lackawanna and Western, Delaware and Hudson, New York, Susquehanna and Western and Pennsylvania Coal Company were able to greatly increase their business, and to secure a much higher average of prices for their coal than they have been able to do for years past. This result might not have been attained had it not been for the extraordinary growth of the demand for anthracite coal. As the manufacturing industries of the country have not been over-active this year, it is clear that the demand has been for household use, and is only proportionate to the enormous growth of the country and the steady cheapening of the cost of transportation, especially upon the Western and Southern railroads. Whether as good results will be secured in 1889 is an open question. Without strikes the competition will be greater, and it is quite probable that the openness of this winter, so far experienced, will so greatly reduce the consumption of coal as to leave the market crowded until next summer, reducing both the demand for, and the prices of, coal. However, this will have but a temporary influence, and, although worthy of the attention of the speculator, it should not disturb the investor, for it is clear that the demand for anthracite coal is rapidly approaching the point where the companies will find difficulty in supplying it, and at the same time keep within the lines of wisdom and economy. When this point is reached, the natural tendency of the leading anthracite coal stocks will be to disappear from the field of active speculation and find their way to the strong boxes of investors. Of course, the speculative managers will delay that day as much as possible."

The production of anthracite coal is controlled by an arrangement, entered into between the large producing and transportation companies, the objects of which are to protect the mutual interests of the producers and shippers, to restrict the product of the mines to the demands of the market, and, at the same time, to obtain a fair and reasonable price for the product, which will insure fair wages to the mining labor and a fair profit to operators and shippers.

The total product of coal in the anthracite region of Pennsylvania during 1888, amounting to 43,922,897 short tons, or 39,216,873 long tons, can certainly be considered phenomenal when the statistical history of these coal-fields for the past 15 years is reviewed; it far exceeded the most sanguine expectations entertained by the best-informed operators and shippers. The year 1888 may be said to have been the most successful year in the history of anthracite mining in Pennsylvania, which has now become the greatest coal-mining district in the United States.

The compactness of the region in which these coal-beds are mined, which does not exceed 325 square miles in area (the area of the entire coal-region being only 470 square miles), and the nearness of the region to the principal coal-markets and most populous portions of the State, containing the largest and most profitable manufacturing districts, may be looked upon as the great causes which have produced this phenomenal development.

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The average price received by the operators for their coal during 1888 was remunerative, and higher than during the previous year. At the beginning of 1888 the strikes and other labor disturbances in the Lehigh and Schuylkill mining regions, which were protracted into the month of March, caused much suffering and great loss of wages to the miners, as well as a large loss of tonnage to the railroads carrying coal from these two regions.

With the exception of the Philadelphia and Reading Railroad Company, the anthracite transportation companies all shipped more coal to market in 1888 than during the previous year, the aggregate increase in shipments amounting to 3,504,700 tons. Nearly 70 per cent. of this increase was from the northern coal-field of the Wyoming and Lackawanna valleys, and known to the coal-trade generally as the Wyoming region. The greatest increase in the shipments of any one company was in those of the Central Railroad of New Jersey, the aggregate increase for the year being 889,420 tons. The only decrease for the year was in the shipments of the Philadelphia and Reading Coal and Iron Company, which amounted in the aggregate to 380,156 tons.

The stock of coal on hand at tide-water shipping points at the beginning of 1888 was 130,977 tons. During the month of January there were 12,379 tons more produced than during the same month in the previous year, and there was a falling off in stock of coal during the same month of 35,809 tons. These facts show an exceptional condition in the coal-trade at the beginning of 1888; a condition explained by the existence of labor troubles, especially in the Lehigh region. The total product of the Lehigh Valley Railroad during the month was decreased 111,597 tons below that of the same month in the previous year; and there was an exceptional decrease during the month in the shipments of the Philadelphia and Reading Company, amounting to 405,918 tons, a decrease also due to labor troubles.

A notable occurrence at the opening of the year was the closure of the receiverships of the Philadelphia and Reading Railroad Company, and of the Philadelphia and Reading Coal and Iron Company, which had been in force from June, 1884. The placing of the Philadelphia and Reading Companies in strong hands opened a better basis for the advancement of its business than had existed since the acquirement by the Coal and Iron Company of its vast coal estates.

The product of the seven bituminous mining districts which send

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coal to the Atlantic seaboard is controlled by the Seaboard Association. The allotment of the percentage of product of each district was agreed upon for 1889, in the early part of March The proportion of each district of the output of the year, which it is estimated will amount to 5,500,000 tons, under the agreement is as follows:

	1888. Per Cent.	1889. Per Cent.
1. Cumberland, Maryland,	30⅛	30⅛
2. Clearfield, Pennsylvania,	30⅛	30⅛
3. Pocahontas, Virginia (N. & W. R.R.),	14⅛	14
4. New River, West Virginia (C. & O. R.R.),	11	11
5. Beech Creek, Pennsylvania,	7½	7½
6. West Virginia Central Railroad,	5½	5½
7. Reynoldsville, Pennsylvania,	1½	1½
Total,	100	100

The governing board is composed of nine members appointed by the operators in the different districts. The Clearfield and Cumberland districts each appoint two members, and the other districts one each. Mr. Galloway C. Morris, of Philadelphia, has been elected chief officer or commissioner.

The board of control determines, from time to time, the quantity of coal to be produced, or, in other words, makes an estimate of what the market will take, so that the production will not materially exceed the consumption. Prices, which are fixed for the opening of the current year at $2.60 f. o. b. at Norfolk, Newport News, Georgetown, Baltimore, and Philadelphia, and $3.50 a ton alongside in New York, may be changed by the board on a majority vote of five of the regions.

The agreement covers the field north of the Chesapeake and the Delaware, and provides that in this field no member of the association shall have the right to deliver coal by all-rail into Connecticut or Rhode Island, or in certain territory in eastern Massachusetts, or to allow rebates to customers or to guarantee freight rates. All sales are to be made at tidewater at the prices fixed by the board.

Each month each producer is required to pay to the chief commissioner 20 cents a ton on his shipments of the previous month, and this money remains as a guarantee fund until three months' payments have accumulated, it being estimated that three months' payments from all the companies will amount to nearly $100,000. Then, upon the payment of the fourth month's assessment, the sum con-

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tributed in the first month is returned, and so on indefinitely during the term of the agreement.

For violation by overshipment above the amount allotted any company by the board, a penalty of 20 cents a ton on the number of tons involved is exacted, and is drawn from the guarantee fund of the offender by the commissioner. The penalties go into a compensation fund, out of which sums are given to those who have not shipped their allotment, any possible surplus being afterward distributed among the other concerns. For other violations of the agreement the commissioner assesses such damages as seem warranted, and deducts this amount from the deposit of the company so assessed.

It is, of course, understood that the penalties of this agreement cannot be enforced in a court of law; but the good faith and mutual interests of the several parties to the agreement, reinforced by the pecuniary penalty for violation of its terms, are relied upon to make the plan effective.

The operators in each region distribute the total allotment for the individual region among themselves.

The agreement runs for three years, from March 1, 1889, to March 1, 1892.

The general prosperity of the coal-trade of Maryland during the year 1888 is reflected in the prices of the stocks of the producing companies. In the case of the Consolidated Company the shares, during the latter part of the year, were largely bought by investors who seemed to be satisfied as to their value, and the return in the form of stock dividends which they could expect on the investment.

The George's Creek Company during the year increased its rate of dividend from 4 to 6 per cent. per annum, and the market-value of the stock has been higher than for several years ($105 being bid, with holders standing firm at $110).

The dividend of the Atlantic Coal Company was reduced during the year about 30 per cent. from what it was during the previous year. The reason for this, however, is stated to be that, although the earnings of the company were quite sufficient to pay the usual rate of dividend, a portion of them was diverted to a sinking-fund established for the purpose of buying new lands.

In the case of the Big Vein Company great fluctuations took place in the value of its securities, evidently caused by the exhaustion of the coal-lands of the company and the purchase of new lands.

The bituminous coal-trade of Pennsylvania during 1888 was an

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exceptional one in many respects, and calls for special comment. Although the increase in the total product of the mines during 1888, which was 2,246,871 short tons, was only a little over one-half of the increase in the product of the mines during 1887 over that of 1886 (4,398,725 short tons), yet the difference between the two years is probably not as great as the figures at our command would indicate.

During the past few years a revolution in their methods of business may be said to have been accomplished by the large coal-consumers, which has resulted in their purchasing coal, not on account of the reputation (possibly unfounded) as a desirable fuel which special coals may have attained, but from actual tests made by consumers themselves as to the relative efficiency of different coals for domestic consumption in house-furnaces, stoves and grates, for manufacturing consumption in the generation of steam, and for the manufacture of gas; the selling price of each coal being considered in conjunction with its actual efficiency, and those coals being given the preference which would give the consumer a unit of efficiency at the lowest price.

This has been more particularly the case since the extensive introduction of natural gas throughout western Pennsylvania, which has very materially affected the local consumption of coal, the Pennsylvania operators being forced to seek new markets for their product. It is estimated that the natural-gas companies of Pittsburgh are now supplying about 28,000 domestic consumers and 900 manufacturing consumers with from 400,000,000 to 600,000,000 cubic feet of natural gas per day, varying with the condition of the weather, which affects all consumers, and with the activity of manufactures. I have estimated that in the vicinity of Pittsburgh alone there was consumed during 1888 a quantity of natural gas, to replace which, if the consumers had to depend exclusively upon coal, would have required during the year about 8,500,000 tons. If to this be added the amount of coal which would be taken by consumers in other parts of the State, who now depend upon natural gas as a fuel, the amount would be increased to at least 10,000,000 tons. It must be remembered, however, that this tonnage does not represent the actual amount of coal which has been displaced by gas, for new enterprises in large number have been started in western Pennsylvania since the introduction of natural gas became general, and householders are keeping their houses more generally heated, and at a very much higher temperature, than formerly, when coal

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had to be depended upon. Still, at the same time, the amount of coal which has actually been displaced by the use of natural gas has been very great, and the coal producers, being forced out of the local markets, have been compelled to seek more distant consumers. This has naturally drawn the attention of the coal-trade in other States to the greater efficacy of the Pennsylvania coals, especially those mined in the Pittsburgh region, over the coals mined in the immediate vicinity of consumers in other States, and a greater distribution of Pennsylvania coal to distant points has been made than ever before known in the history of the United States coal-trade.

In 1884 and 1885, when the Pennsylvania coal-operators were forced by natural gas into outside markets, the situation was viewed with great alarm by the coal-trade, and, in order to open up new markets, Pennsylvania coal was sold at very much lower prices at the mines than it had formerly commanded when so largely taken by local consumers; in consequence the price of Pennsylvania coal in 1885 and 1886 fell. The average price of coal at the mines in the Pittsburgh region, during 1886, ranged from 88 to 92½ cents per ton at the mines; and the average price at the mines throughout the State was only 80 cents. As soon, however, as the distant consumers began to appreciate the greater efficacy of the Pennsylvania coals as compared with others, particularly those mined in Ohio, Indiana, Michigan, Illinois and parts of Tennessee, Kentucky and Missouri, the Pennsylvania operators were enabled to command a better price for their product, so that during 1887 the average price which the Pennsylvania coals commanded at the mines was 10 cents a ton more than during the previous year, and it was 5 cents a ton more during 1888 than during 1887. Part of this increase in price was due to the general improvement in business, with an accompanying improvement in the coal-trade.

The increased popularity, among distant consumers, attained by Pennsylvania coal, particularly that of the Pittsburgh region, is more noticeable in the increased tonnage of the mines than in the increased average price. As an illustration of this fact, in Allegheny county, surrounding Pittsburgh, where more natural gas is consumed than in any other one locality in the United States, and where the local consumption of coal from the surrounding mines has been, of course, more directly affected, the increase in the product of the coal-mines has been greater than in any other mining district in the country. During 1887 the mines of this county produced 4,680,924 tons, an increase over the previous year of 478,838

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tons. During 1888 the same mines (practically the same, there was only one new operation started in the county during 1888) produced 5,575,505 tons, an aggregate increase over 1887 of 894,581 tons.

The situation of the coal-trade in the Mississippi Valley, and generally throughout the West, is a still more complex problem in detail, although much less coal is handled than on the Atlantic seaboard.

West of Pennsylvania, the most important coal-mining districts influencing the coal-trade are those of the State of Illinois. In 1887 the Indiana, Pennsylvania, Ohio and West Virginia bituminous coals made large inroads into the soft-coal trade of Chicago and the Northwest, which the Illinois operators regard as belonging to them. During 1888 the Illinois operators determined to regain the position which they formerly held by means of extremely low prices to the trade and by a reduction in wages, and this was in a large measure accomplished, the result being that the product of the Illinois mines during 1888 increased at a greater rate than the product of the Ohio mines, which enters the same territory, and occasioned a falling off in the product of the mines of Indiana. The effect of the action of the Illinois operators upon the product of the Pennsylvania and West Virginia mines was not noticeable.

The distribution of coal from Duluth Harbor, Superior and Saint Louis Bays, in the Northwest, is rapidly increasing. The editor of the *Pioneer Express*, in referring to the growth of the Duluth coal-trade, says:

"More coal is now unloaded in Duluth Harbor, Superior and Saint Louis Bays than in any other port, if not in any other two ports, on the chain of the great lakes. The great development of Duluth's coal-trade is due to the great growth of the Northwest and the requirements of its people, railroads and manufactures, and also to the increase of transportation facilities to the interior and the advantage of cheaper coal-rates by lake than Lake Michigan ports enjoy. The first reason would not account for the large increase of this year's trade, which has been much greater than the growth of the Northwest country tributary to the head of the lake. This fact shows that Duluth is now supplying coal to considerable territory formerly dependent wholly on Chicago and Milwaukee, or the Iowa coal-fields. Her present territory for coal sales includes all of Minnesota and nearly all of Dakota, Montana, as far as Livingston, and parts of Wisconsin, Iowa and Nebraska. If there were fair rail-rates for coal from Duluth to interior points, as compared with other ports, there is no question but that her coal business would be largely increased over its present size, and at a much more rapid rate, until all the coal required in the Northwest, west of central Wisconsin, would come from the head of Lake Superior. As it is now, the coal-rates from Duluth to Fargo and other common points are as high or higher than the rates from Milwaukee or Chicago. In other words, it costs the same or more to ship a car-load from Duluth docks, 285

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miles, as from Chicago docks, 700 miles. This is a discrimination which is too unfair to last. Though rail-rates on coal are higher from Duluth than Lake Michigan ports, it is the reverse so far as lake freights are concerned. The past season's coal freights from Lake Erie points to Duluth ranged from 50 to 85 cents, with the average less than 70 cents, or from 25 to 40 cents less than the rates, at the same time, to Chicago, Milwaukee and Green Bay."

In reference to the coal-trade of the Southern States, and the area between the Rocky Mountains and the Mississippi Valley, little can be said other than that the consumption of coal is rapidly increasing, depending upon the industrial development of the districts, and the more favorable prices that are generally secured from increased competition and increased shipping facilities.

Along the Pacific slope for the past two years the price of coal for household use has ruled high, the highest price being reached in 1888, when at several points Wellington coal, from British Columbia, sold to the trade at $16 per ton. This high price of a favorite coal led consumers to take other and less expensive coals, and naturally the increased consumption of these other coals caused the prices to be advanced. The high price invited the importation by rail of coal from the Rocky Mountain mines to San Francisco and to interior central cities, and of coal from New Mexico into the southern part of California. In addition, it stimulated the out-put of older mines along the coast, and the opening up of several new collieries. The product of the new California mines played no important *rôle* in the general coal-trade of the Pacific slope, still California produced 95,000 tons of coal during 1888 as against 50,000 tons during the previous year.

December 1, 1888, the price of Wellington coal was unexpectedly dropped to $12 per ton, compelling a corresponding reduction in price by other coal-mining companies, which, later, was followed by a further drop. This last decline shut off the importations of Rocky Mountain coals, since they cost, delivered, $10 per ton. The mild character of the winter, combined with the increased use of manufactured gas for fuel, and oil-stoves, had quite a decided effect on the consumption of house-coal, which has been much less than during the same time in 1887, and this, too, in the face of an increased product. The consumption of steam-coal on the Pacific slope has increased to large proportions, and to this, no doubt, is due the ease with which coal-values were maintained in 1887 and 1888. The increase in the number of steam-vessels, locomotives, and manufactories has been fully 50 per cent. during the past two years, creating

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a corresponding consumption of steam-coal. Another important reason why the price of steam-coals has been more easily maintained during 1888 was a three months' strike of miners in the Australian collieries.

Coal-Miners' Wages in the United States in 1888.

The wages paid for mining coal and for outside labor around coal-mines, and directly connected with the handling of wrought coal from the top of the mine-shaft or the mouth of the mine-drift to railroad cars on which the coal is shipped to market, varies greatly throughout the United States, but is closely connected with peculiar and local conditions which characterize each mining district. These wages are the subject of an almost constant dispute between the employer and the employee. Primarily they are based upon the market-price received for the product of the mine, since, except under very peculiar and purely local conditions, and for very limited periods, mines are not kept in active operation at a direct loss to the operator.

The actual wages paid to the laborer, when the mines are kept in operation at any profit, however small, to the operator, are sometimes determined in special cases by a co-operative action between individual employers and employees, but more generally by co-operate action between all the operators and employees in a district where mining is carried on under essentially the same conditions. Frequently, however, the wages are the subject of demand by the employee upon the employer, and when these demands are not acceded to by the latter the refusal is often made the object of a strike by the former, resulting in the stopping of all active mining operations for longer or shorter periods, entailing great pecuniary loss both on the operator and on the laborer. The inevitable result of such a strike is that while the operator loses much money during these periods, in case there is legitimate demand for his coal, yet the actual personal suffering is at all times infinitely greater to the miner and his family, whose distress for the want of proper food and clothing, and even at times shelter from the weather, is frequently heart-rending.

A strike among miners often takes place and is very foolishly instigated by the miner himself at periods of temporary depression in the coal-trade, and in these instances the opportunity afforded the operator of closing his mine is eagerly embraced as a means of preventing possible loss to him in the conduct of his business.

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A lack of a proper understanding between the employer and his men, and a failure to co-operate in the mutual interests of each, has led to the organization, in almost every district, of local miners' labor-unions, or more wide-spread amalgamated associations, which have endeavored to establish an absolute fixed price for all labor interested in mining without a due regard to the welfare of the operator. This has forced the operators, as a counter-action, to pool their interests, so that to-day in all the prominent coal-mining districts of the country the determination of a scale of mining-wages is a matter of very delicate adjustment between the individual and corporate operators and the laborers employed in and about their mines. The scale of wages which was in force during 1888 varied considerably in the different States and Territories, and in most cases was dependent upon many conditions purely local in each mining district.

It is not proposed here to review the action of conventions which were held during the year, either by the coal-operators or the mining laborers, or to refer to the practical results brought about by conferences between employers and employed, but merely to give the number of laborers employed in each State and Territory, the tons of coal they produced, the price which the coal commanded at the mouth of the mines, and the wages paid the miners. These facts are exhibited in the table on page 137. The wages paid to miners and laborers in a number of States and Territories varied greatly throughout the year; the figures given in the table may be taken as the average for the year.

The column headed "inside and outside labor employed" includes only the number of miners and laborers engaged in removing the coal from the bed, and placing it on railroad cars and wagons at the mouth of the mine for shipment to market, and are the daily average for the year of 300 working days.

The number of all miners and laborers employed in the different districts for the removal of 10,000 tons of coal within the space of one year varies greatly. In the first place, it depends upon the number of days worked during the year. Most of the coal-mines throughout the United States worked, during 1888, more than 250 days, and a large number worked practically every day, except Sundays and the holidays. The number of inside and outside laborers employed is the average number employed for those days during the year when the average daily tonnage of coal was produced, this average being ascertained by dividing the total yearly product by the number of days worked. In very few instances were any mines worked for so

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States and Territories.	Short Tons Produced.	Value per Ton at Mines.	Inside and Outside Labor Employed.	Wages Paid for Mining Noted in Cents.
Pennsylvania Anthracite	43,922,897	$1.95	101,000	85 per car of two gross tons, subject to adjustment on sliding basis of price of coal.
Bituminous	33,796,727	0.95	62,000	P. R.R. mines, 69 net ton; Clearfield, 45 net ton; Allegheny Mtn., 40 net ton; Connellsville, 89 net ton; Myersdale, 40 net ton; Monongahela, 3 bu. (76 lbs.); Reynoldsville, 65 net ton.
Illinois	14,655,188	1.12	29,410	Mt. Olive, 50 net ton; Peoria, 70 net ton; Spring Valley, 70 net ton; Wilmington, 80 net ton.
Ohio	10,910,946	0.98	27,000	Hocking Valley, 60 net ton; generally 45 to 65 net ton.
West Virginia	5,498,800	1.10	9,700	Fairmount, and B. & O. R.R. mines, 35 gross ton; Kanawha River, 2½ to 3 bu. (80 lbs.); Elk Garden, 50 gross ton.
Iowa	4,952,440	1.30	9,600	Angus, 75 gross ton; Flagler, 65 to 75 gross ton; Fort Dodge, 4 bu.; Ottumwa, 75 gross ton; What Cheer, 75 gross ton.
Missouri	3,909,967	1.50	8,800	50 to 60 net ton.
Maryland	3,479,470	0.95	5,950	50 gross ton.
Indiana	3,140,979	1.40	7,403	.....
Alabama	2,900,000	1.15	6,900	Generally 40 to 65 net ton; Montevallo, 100 net ton, 2 ft. bed.
Kentucky	2,570,000	1.20	6,243	Eastern mines, 75 net ton; Western mines, 62½ net ton.
Colorado	2,185,477	2.23	5,575	Boulder, 82½ net ton; Crested Butte, bituminous, 75 gross ton; Baldwin, 100 net ton
Tennessee	1,967,297	1.10	4,800	50 to 65 net ton.
Kansas	1,850,000	1.50	3,600	Average, 5½ bu. (80 lbs.); Leavenworth, 4 bu. (80 lbs.).
Wyoming	1,481,540	3.25	2,459	Lump coal, 60 net ton, and 40 slack coal.
Washington	1,215,750	3.00	3,000	350 per day by measure and contract.
Virginia	1,073,000	1.00	2,400	Pocahontas, 40 net ton; Eastern mines, 50 to 75 net ton.
Indian Territory	761,986	1.95	1,700	80 net ton.
New Mexico	626,665	3.25	1,500	Average per day, 250 to 400, paid on ton basis.
Arkansas	276,871	1.50	650	80 to 125 net ton; average, 87 net ton.
Utah	258,961	2.10	500	Varies from 80 net ton to about 300 for miners to 200 for laborers per day.
Georgia	180,000	1.50	500	Convict miners, contracted.
California	95,000	4.00		Adjusted on basis of about 250 daily earnings.
Texas	90,000	2.05		Varies greatly, ranges from 45 to 130 net ton; T. & P. C. Co., 125 net ton.
Michigan	81,407	1.60		125 to 250 per day basis; Bennett S. P. Co., 75 net ton.
Oregon	75,000	3.00	1,070	Coal Bay, 100 net ton.
Montana	41,467	3.75	(Estimated.)	Based on daily earnings, 300 to 450.
Dakota	34,000	1.75		Based on daily earnings, 250 net ton.
Rhode Island	4,000	2.75		Varies greatly, average earnings about 190 net ton.
Nebraska	1,500	2.25		Unknown.
Idaho	400	4.50		Prospecting labor exclusively.
Total, exclusive of colliery consumption	142,037,735	.....	Labor in mining coal consumed at collieries included above.	
Colliery consumption Pennsylvania anthracite	2,696,667	.....		
Bituminous in all States and Territories	3,925,000	.....		Same wages paid in each coal-field for mining coal consumed at colliery as for mining commercial coal.
Grand totals	148,659,402	.....	302,860	.....

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short a time during the year that the miners were only temporarily located in the vicinity of their mines; as a rule, they were permanent residents during the year, so that the total of 302,860 men employed may be taken, with their families, to represent the permanent population around about the coal-mines of the United States.

In the second place, the number of miners employed depends upon local conditions peculiar to each mining district. Some of these conditions are the thickness of the coal-bed, the hardness or softness of the coal, the existence or absence of thin layers of slate or clay included within the body of the bed, which necessarily retards the mining, the care which is necessary to remove the slate and bony coal from the market product, and the necessary handling of the coal between the mine-mouth and the railroad shipping car. Much also depends on whether the coal-bed is comparatively horizontal, or dips at various angles, owing to an irregular and complex geological structure; whether the bed is mined from self-draining drifts and tunnels, or from the bottom of slopes and shafts, from which the coal has to be hauled and the mine-water pumped; whether the coal has to undergo any preparation after it is taken from the mine, before it is in condition to ship to market; whether it has to be washed, as some of the Pennsylvania anthracites and Southern bituminous coals have to be; whether it has to be broken down in commercial sizes, as is the case with all anthracites. Many other conditions might be named which directly influence, though to a small degree, the number of laborers which are required for the mining of a definite quantity of coal.

Taking 10,000 tons of coal mined per year as a unit of product, I find that the number of inside and outside laborers taken together, which are employed about the mines, varies from 17 as a minimum in the State of Maryland, to 26 as a maximum in the State of California. In West Virginia the number is 18, in the Pennsylvania bituminous region 19, in Kansas and Indian Territory 20, in Illinois 21, and in the Pennsylvania anthracite region 23. In all the other States and Territories in which coal is produced, the number of men employed for the production of 10,000 tons of coal per annum varies from 22 to 25. The peculiarities in the character of the coal-bed, which have already been referred to, influence not only the number of miners employed in mining a unit of product, but the rate of wages paid, it being obviously just that where one miner can mine twice as much coal in one district as another equally competent miner can mine in another district, that there should be a

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marked discrimination in the amount of wages paid to each. While it is of great interest to coal operators and to the mining laborer to be informed as to the rate of wages paid in different districts, yet, unless the local conditions peculiar to each district are fully understood, no just appreciation can be formed as to the absolute daily profit realized by labor in each locality.

In addition to the conditions which affect this question already noted, it is important to bear in mind the different purchasing value of a dollar in the different mining communities, in any estimates made of the actual earnings of the miners and other laborers in and about the coal mines.

THE ORE-DEPOSITS OF RED MOUNTAIN, OURAY COUNTY, COLORADO.

BY T. B. SCHWARZ, RED MOUNTAIN, COLO.

(Colorado Meeting, June, 1889.)

THE Red Mountain district, in Ouray County, Colorado, has been already referred to in the *Transactions* of the Institute, and notably in papers by Mr. T. B. Comstock, on "The Geology and Vein-Structure of Southwestern Colorado" (*Trans.*, xv., 218), and on "Hot-Spring Formations in Red Mountain District, Colorado—A Reply to the Criticisms of Mr. Emmons" (*Trans.*, xvii., 261), and in the paper of Mr. S. F. Emmons, on "Structural Relations of Ore-Deposits" (*Trans.*, xvi., 804).

The district has attracted much attention from those interested in geology, because it seems to present a variety of ore-deposits rarely, if ever, met with elsewhere, and one which cannot be described by such well-understood terms as fissure-veins, bedded-deposits, or contact-formation.

It will be the scope of this paper to note briefly, from the writer's experience in developing the Yankee Girl, Guston and other Red Mountain properties,—first, observed facts as to the ore occurrence and formation, and second, certain unsolved geological questions.

The Red Mountain district proper occupies the headwaters of Red creek, from Ironton to the Divide, a distance of 5 miles. Its leading properties occur on the west slope of Red Mountain. The mode

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of occurrence of the ore of Red Mountain, in its entire extent from Ironton to Silverton, 10 miles distant, is, however, identical with that of the limited district to which this paper refers. Outside of this district the development is slight. Its mean elevation is about 10,800 feet above sea-level, while the adjoining peaks rise to about 13,000 feet. It is in the heart of a very extensive area of eruptive rocks, andesites, trachytes, and breccias, in which occur the most productive camps of the San Juan country, such as Marshall Basin, Mt. Sneffles, Mineral Point, Lake City, and Silverton, all within a radius of 15 to 20 miles.

The topography of the district itself is characteristic, and indicates something of its past history. It is marked by bare, ragged cliffs, and red patches of oxidized material on the upper mountain sides, while below are considerable areas of heavy slide-rock, carrying large detached masses. Between these areas of detritus occur benches and terraces covered with shallow soil, and diversified here and there by knolls or mounds of hard, porous, siliceous rocks, with a reddish hue or stain. They sometimes rise above the surrounding bench in a broken cliff, with a sloping base of rock fragments; or again, barely show an outcropping of quartz. The basin presents a very rough, broken surface, showing everywhere the effects of heavy erosion, and of an active period of metamorphic action. Good water is not to be found. Iron springs abound, and, in depth, the mine-water becomes all that is objectionable, carrying 24 grains of SO_3 to 1 U. S. gallon, and readily disposing of a $\frac{1}{4}$-inch thick wrought-iron column-pipe in four weeks' time.

The ores of the district may be divided into two groups, viz.:

A. The secondary, or oxidized ores.

B. The primary, or unoxidized ores.

In some properties ores of both characters occur, the one beginning where the other terminates.

The Secondary Ores.

The secondary ores occur above a former water-line, either attached to the walls of caves, as broken, detached masses, or as a bed of clayey mud or sand, more or less completely filling the caves. The cave formation is identified with the massive outcroppings or knolls of silicified andesite, ordinarily termed "quartz." These knolls rise from 25 to 200 feet above the surrounding surface, and while sometimes quite conical in appearance, they more generally have a greater length than width, in some instances being 400 to

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500 feet long, by 200 feet wide. They present a rough mass of quartz, cut up by cross fractures, and showing small vugs and cavities on the exposed cliff-faces. The ore-bearing caves, which ramify throughout the mass, generally come to the surface along the cliff-base, where they are partially or wholly covered by slide. In size the caves vary up to chambers of 50 feet in diameter, which are connected by irregular rounded passages, branching out towards the surface, but diminishing and coming together in depth.

The ores are mainly carbonates of lead and of iron, together with the iron oxides, lead sulphates, and arsenates. Kaolinite occurs in considerable quantities, and zinc-blende is common. The latter occurs in botryoidal masses, consisting of nearly concentric fibrous layers, and is usually found detached from cave-walls. Galena also occurs, but generally as the core of an oxidized mass. Such are the ores of the National Belle, Grand Prize, and Vanderbilt mines, while other properties near by, viz., those in the Enterprise group, omit the lead minerals and carry the oxide and sulphide of bismuth. From these latter ores, sorted to a grade of 25 to 50 per cent. of metallic bismuth, several hundred pounds of the metal have been reduced by Messrs. Mentzell & Kellar, of Durango.

The following are the prominent features of the secondary ores:

1. The secondary ores are richer than the sulphide ores occurring below them.

2. The ores of adjoining or connecting caves are sometimes greatly different in grade.

3. In some cases the formation of the caves along fracture or cleavage planes is evident, but in others all traces of such planes are quite obliterated.

4. The cave-walls are a porous, sandy quartz, the sand from the disintegration of which forms part of the cave-filling.

5. The line of change from oxidized to unoxidized ores, or the former water-level, is very marked. It varies as much as 100 feet in elevation in properties within 1000 feet of each other, rising to the south and west. The quartz outcrop rarely rises more than 200 feet above it.

6. In isolated cases may be found masses of the unoxidized ore, the enargite, above the line of change, in the vicinity of the secondary ores.

The formation of these ores is readily accounted for in the oxidizing and dissolving properties of surface-waters, which, moving along fracture planes, took into solution the original sulphide-ores and

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portions of the adjoining rock, and deposited new ore-bodies in the resulting caves.

The Primary Ores.

The bulk of the Red Mountain product has consisted of the primary ores. Although large amounts of the carbonate ores have been shipped from such properties as the National Belle and Vanderbilt, their depth is soon exhausted and sulphide ores are reached. The latter begin where the cave-ores cease, and in many properties, such as the Yankee Girl, Hudson, Guston, and Silver Bell, in which the cave-formation has been removed by erosion, or never existed, they outcrop at or near the surface. They consist of a great variety of the sulphides and sulpharsenides of the metals, and are productive of the following minerals in quantity, viz.: Enargite, galena, chalcopyrite, pyrite, erubescite, bismuthinite, gray copper, and stromeyerite. Among the metallic minerals of rarer occurrence are: Silver glance, polybasite, tennantite, proustite, and others not yet determined. Associated with these ores are: Rhodonite, gypsum, heavy spar, and kaolinite.

The miner recognizes underground in this district three essentially distinct varieties of rock, viz.:

1. A hard, fine-grained, greenish rock, provisionally called diorite.
2. A soft, white andesite.
3. A breccia.

The main features of the ore-formation may be briefly stated, as follows:

1. The ores occur as "chimneys" so-called, having in some cases an elliptical or a circular cross-section, but more generally long in proportion to their width. The greatest length of ore-body so far observed has been about 60 feet.

2. The immediate envelope of the ore-chimneys is "quartz," which is sometimes of considerable extent, while the whole is enclosed in an area of greater or less extent of andesite.

3. The andesite is metamorphosed in proportion to its nearness to the ore-chimney, showing more kaolin and quartz, and less of its original structure. It thus merges into a fine-grained, brown or grayish quartz, sometimes very hard and flinty, and again quite porous. In depth the quartz matrix becomes less characteristic.

4. The areas of andesite are separated by areas of block-like diorite, which, though varying in texture and hardness, is unaltered and barren of ore.

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5. The division between the diorite and andesite is generally sharp and well-defined, but in places the one merges into the other.

6. The diorite "wall," or line between the two rocks, is exceedingly variable, maintaining no regular course, but is undulatory both in vertical and horizontal directions.

7. The andesite areas are more extensive below the surface than at their outcropping.

8. In the case of six chimneys occurring on the Yankee Girl, Guston, and Silver Bell properties, being the only ones on which depth has been obtained, a marked increase in the silver-content of the ores occurred from the surface down to 300 to 400 feet of depth.

9. Changes in character of ore with depth are noticeable in several chimneys, but notably so in the main Yankee Girl chimney. In this case the distinctive minerals, in order of depth, have been galena, gray copper, stromeyerite, and bornite. In those chimneys in which enargite has been the surface-ore no depth has yet been obtained.

10. The chimneys frequently change their pitch, sometimes quite suddenly. Any one chimney may recede from a given vertical line for a considerable distance, and then approach it again. A jump of 15 to 25 feet along some horizontal plane is not infrequent, rendering it difficult to locate the chimneys on succeeding levels.

11. Every chimney is connected with one or two cleavage or fracture planes, which bound a portion of the ore-body, while on the other sides, the ore merges into the quartz or andesite. These planes usually show striations, slickensides, and more or less gouge material, but are not of great persistence laterally.

12. Horizontal planes occur of great strength, which have indirectly had marked influence on the ore-bodies, changes in character occurring at such horizons.

13. The areas of breccia are sometimes heavily mineralized, carrying galena and gray copper in the cementing magma.

14. In the Guston and Yankee Girl the increase in the amount of the copper-ores with depth is notable, as is also the fact that the silver is confined mainly to such ores.

Geological Features.

Mr. Emmons, in his paper on the "Structural Relations of Ore-Deposits," referring to this district, says of the knolls (*Trans.*, xvi., p. 809):—"They are simply portions of the andesitic country-rock from which acid waters have removed the basic constituents, per-

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haps depositing a certain amount of silica in their place; the resulting quartzose masses, offering greater resistance to the disintegrating effect of atmospheric agents and to erosion than the surrounding rocks, have been left as mound-like ridges protruding above the general surface, more or less independent of the natural drainage channels;" and of the ore-deposits (*Ibid.*, p. 834):—"In the Yankee Girl there are several chimneys in which the ore occurs. They are of elliptical outline, the longer axis corresponding in direction with a main system of fractures running through the region. Although the striated surfaces of these planes show that there has been movement along them, there is but little evidence left of actual brecciation of the country-rock, the ore-solutions having completely replaced the andesitic country-rock between the fracture planes which gave access to them; in places a siliceous skeleton is left, the basic constituents being replaced by vein-materials; in other places a solid body of metallic minerals is found, while the country-rock adjoining the body of pay-ore is impregnated to a considerable distance with low-grade sulphurets."

But little can be added to this general history of the formation, until such time as the section can be studied comprehensively by a trained corps of geologists, whose field-work can be aided by the laboratory. The facts already cited indicate strongly that the mineral solutions found access into the fracture planes along which the chimneys are formed, largely through the horizontal planes—possibly bedding-planes—and that their mineral contents were mainly obtained from the eruptive andesite of the district. At each horizontal plane it is probable that a fresh current of mineral solution reached the chimney, and, by a change in the character of the main current of ascending waters, accounted for the change in the nature of the ore-deposition. Mr. Emmons notes that the longer axis of the ore-bodies corresponds "in direction with a main system of fractures running through the region." While this is generally true, there is a marked exception in the case of the main ore-body on the Guston mine, whose course is at right angles thereto.

The relation of the andesite to the rock areas I have provisionally termed diorites, is an undetermined feature. These latter seem to come from the mountain mass to the west, and resemble in many features intrusive dykes, but lack their regularity. Until more light can be thrown on the subject, the theory that the one rock is the result of metamorphic action on the others, seems the most plausible.

That the alteration of the andesite and the deposition of the ore-

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bodies was a simultaneous process, the facts mentioned seemed to leave no room to doubt.

To the practical mining man as well as to the theorist, mineralogist, and geologist, this section is a most interesting one. Chemical and structural geology here have a brilliant field for study. Many new and rare mineral combinations may here be brought to light. The miner is interested in a district which presents more difficulties in following one's ore-body than almost any other, and requires the most careful study of rock-faces.

One of the deepest shafts in the State, that of the Yankee Girl, nearly 1000 feet vertical, is located here, and the grade of the ores from some of the mines has been often subject of comment. The writer has shipped from the Yankee Girl several carloads of ore carrying 1500 to 3000 ounces of silver to the ton, and one lot of some six tons returned 5300 ounces to the ton.

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*THE GEOLOGY AND ORE-DEPOSITS OF IRON HILL, LEADVILLE, COLORADO.**

BY A. A. BLOW, LEADVILLE, COLORADO.

(Colorado Meeting, June, 1889.)

THE productive area of the Leadville mining district immediately adjoining the city of Leadville, which, by its wonderful richness, has attracted the attention of the mining world for the past twelve years, may, with a few exceptions, be considered as divided into seven local mining centers, separated one from another by the topographical features of the country, and designated by common consent as Fryer Hill, Yankee Hill, Breece Hill, Rock Hill, Printer-Boy Hill, Iron Hill, and Carbonate Hill.

Of these seven distinct mining localities, or groups of mines, the central one, Iron Hill, is the one of greatest importance and interest.

It was the scene of the earliest discovery of carbonates, as far back as 1875, and since that time has always maintained its position

* The first part of the present paper is a reproduction in a revised form of a paper by the same author, entitled "The Ore Chutes and Recent Development of Iron Hill, Leadville, Colorado," and published in *The Annual Report of the State School of Mines of Colorado for* 1887.

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as a steady and heavy producer of gold, silver, and lead-ores. Area for area, it may be regarded as the region of the most extensive mining operations, profitable results, and permanent prospects of any portion of the entire Leadville district.

Covering about 200 acres, heretofore unequalled in wealth by any of its associates, it embraces within these limits the rich and productive properties of the Iron Silver Mining Company, the Silver Cord Combination Mining Company, the Iron Hill Consolidated Mining Company, the Louisville and Colorado No. 2 mines, the Smuggler Consolidated Mining Company, the Colonel Sellers mine, the Mikado Mining and Smelting Company, and the A. Y. and Minnie mines.

Since the early day of the primitive horse-whim and windlass, Iron Hill has always been able to boast of the largest and richest mines of the camp, and to-day, with its Mikado, A. Y., and Minnie mines, makes good its claim to the bonanzas of Leadville.

Location and Topographical Features.

Iron Hill is situated about 2½ miles due east of the city of Leadville. It is the second of a series of flat-top steps or bosses, rising easterly from the valley of the Arkansas river, which form the western slope of the Mosquito range. From the flattened summit of Carbonate Hill and Graham Park, Iron Hill rises gradually about 400 feet, attaining at its top an elevation of 11,050 feet above sea level, and then, like Carbonate Hill, extends easterly with a gentle slope into Adelaide Park and the foot of Breece Hill.

With its longer axis north and south, Iron Hill slopes gradually in a northerly direction to Little Stray Horse gulch, a shallow watercourse having its source in Adelaide Park, and dividing it from the long, narrow ridge of East Fryer Hill and its continuation, Yankee Hill; on its southwestern and southern slopes it sweeps boldly around, with increasing steepness, to California gulch, a deep V-shaped valley, eroded several hundred feet below the normal scoring of the hill-surface, and exposing upon its steep flanks the edges of a large portion of the remaining sedimentary strata, together with the outcrop of the enclosed Iron Hill vein.

It was in the bed of California gulch that the carbonate ores of the Leadville mining district were first discovered, in the spring of 1874, by Mr. W. H. Stevens and Mr. A. B. Wood, and this discovery was immediately followed by the location of the entire outcrop of the great Iron Hill vein.

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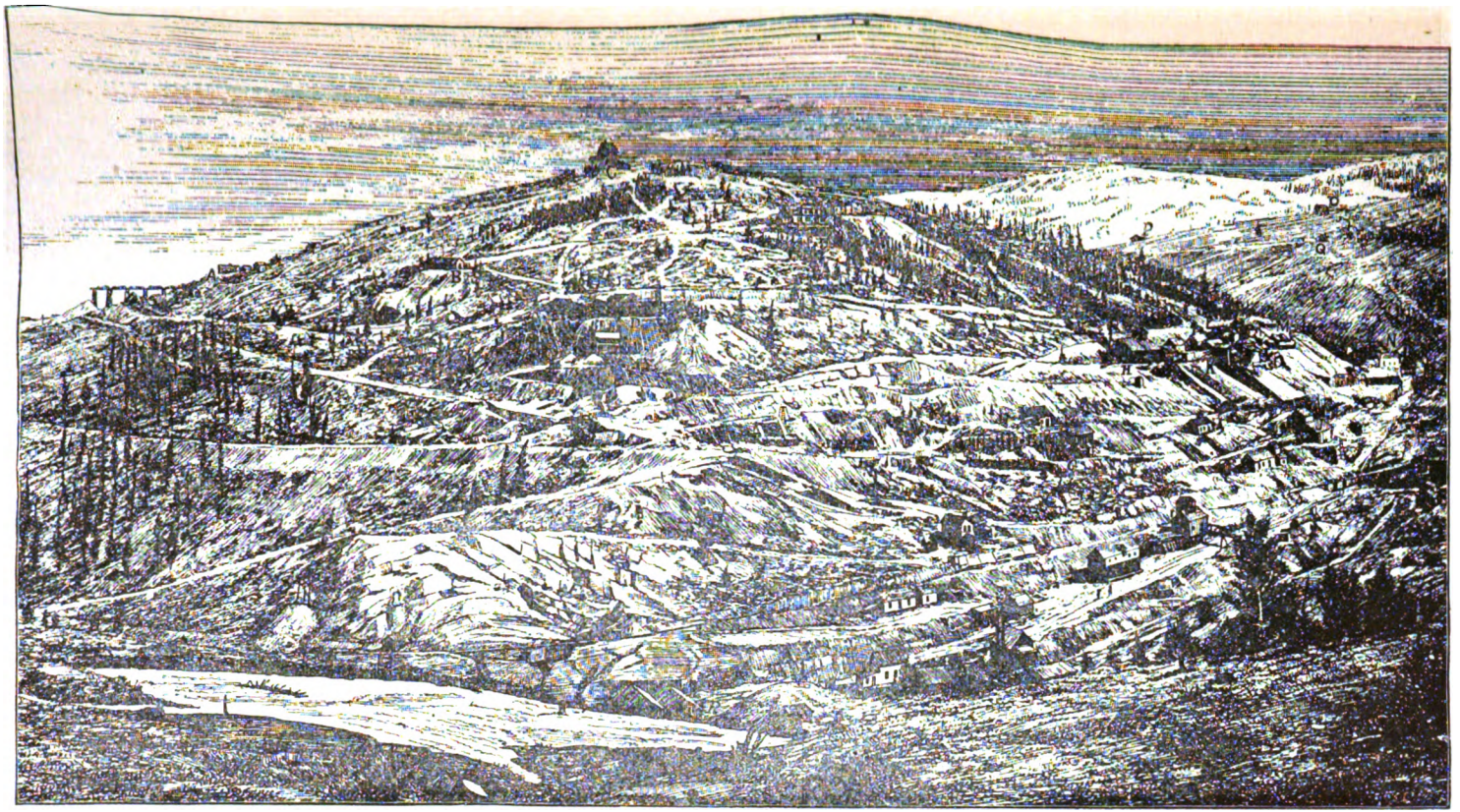

GENERAL VIEW OF IRON HILL, LEADVILLE, COLO.

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In the sketch accompanying this paper a view of the southwestern and southern slopes of Iron Hill is shown, looking from Rock Hill northeast up California gulch. In the map the northern quarter of the hill is not shown, though its general topography can be easily determined from the contour lines, which are drawn at intervals of 10 feet vertical distance, and traced from actual survey.

In the exhaustive monograph of Mr. S. F. Emmons upon "The Geology and Mining Industry of Leadville," probably the most complete and interesting report of the kind ever issued by the United States Geological Survey, the topographical features and general geological structure of Iron Hill have been accurately defined. Since Mr. Emmons's examination of this district in 1882, the comparatively insignificant development then existing in the Iron Hill mines has been increased ten-fold. What was at that time *terra incognita* on the southern and eastern slopes is now the bonanza district of Leadville. The entire thickness of the blue limestone, rather than its upper contact, has been proved to be the true ore-horizon, and, as predicted by Mr. Emmons, the carbonates, with deeper developments, have given place to sulphide ores. Seven years of active exploration have served rather to complicate than to simplify the study of the general structure of this region, though, at the same time, they may be truly said to have confirmed to a most remarkable degree, and, with few exceptions, the skilful deductions and theories advanced by Mr. Emmons from the comparatively meager data then at his disposal. It is therefore not the intention at this time to discuss the generally accepted principles and theories, so ably and clearly presented in Mr. Emmons's work, but rather to describe briefly, and to illustrate more clearly by plan and section, some of the more recent developments of this region, which are to have an important bearing upon its future successful operations. It is hoped that some of these observations may prove of interest to the Institute, and to persons engaged in mining operations.

General Geological Structure.

As to its general geological structure Iron Hill may be considered as a portion of the Cambrian, Silurian, and Carboniferous sedimentary formations, found in the Leadville district, consisting of quartzite and two dolomitic limestones, deposited at the bottom of the palæozoic seas, and resting upon the Archæan granites. Through and between these sedimentary rocks were intruded at different periods, and from different vents or sources, two quartz-porphyries

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or felsites. That the eruption of these porphyries occurred at different times, with a long intervening interval, is shown by the distinctive characteristics of each, by their uniformly varying component parts, and by the intrusion of one under or into the other, with distinct planes of contact and lines of demarcation. Both of these porphyries of Iron Hill, however, are highly siliceous, or quartz-porphyry; they are classed by Mr. Emmons as the most acid of the older porphyries of the district. The oldest or white porphyry, which constitutes the country rock or "block porphyry," chose for its plane of least resistance and intrusion the plane of contact between the Lower Carboniferous blue limestone and the Weber shales and grits, forcing them apart and spreading itself conformably on the upper surface of the blue limestone to a thickness of over 1000 feet.

In this immediate region the white porphyry has never been found in the form of dykes normal to the strata of the sedimentary rocks; though it probably may exist as dykes in localities nearer its source. The contact of the white porphyry with the blue limestone may be regarded as the top of the ore-horizon of Iron Hill, and the entire thickness of the limestone as the vein-rock or lead.

Some time later than the intrusion of the white porphyry the eruption of the gray or dyke porphyry occurred. This porphyritic ejection, though apparently less extensive than that of the white porphyry, is, at present, in its relation to the ore-chutes and channels, and to the folding and faulting of the strata, the most interesting and important subject of study in the entire Leadville mining district. Breaking up through the lower formations under enormous pressure, both from below and from the thousands of feet of superincumbent rock above, it forced itself principally between the lower surface of the blue limestone and the upper strata of the white limestone, seeking the weaker lines of these sedimentary rocks, and cutting across their strata in the form of numerous dykes, from which, again, numerous thin sheets of intercalated porphyry of great horizontal extent spread between their bedding-planes, and also between the contact of the blue limestone and the overlying pre-existing white porphyry.

In this process of forcible intrusion and compression of two bodies into the same space at the same time (a statement rather paradoxical, in a physical sense), the colder and more rigid limestone strata were compressed, crushed, bent, and partly broken, or, more tersely to express it, "buckled up," leaving the lines of disturbance nearer the dykes themselves in the direct relation of cause and effect. It

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was clearly *after* the eruption and intrusion of both of these porphyries, and through their agency, most probably that of the junior, gray, or dyke porphyry, that the ore-bodies of Iron Hill were deposited by metasomatic replacement and substitution of the blue limestone, the ore being brought in mineral-bearing waters and vapors coming up from *below*, and following the course of the gray porphyry.

It is also clear that the mineral-bearing solutions, choosing and following the lines of easiest penetration and substitution in the limestone, gradually deposited their mineral contents along the horizontal planes of contact between the igneous and the sedimentary rocks, and within the broken and crushed zones previously caused by the intrusion of the dykes; and further, that the ore-bodies, whose carbonate form is the result of secondary alteration and realteration, were originally deposited in the form of sulphides.

Following the porphyry intrusion and the subsequent ore deposition, and as a culmination of the dynamic action of pressure and tangential contraction already commenced, came the movements which produced the principal folding and faulting of the Leadville district, and the elevation of the Mosquito range. As is now fully demonstrated from recent developments, the igneous and sedimentary rocks, as well as their enclosed ore-bodies, all partook of this general plication and faulting, the principal or major faults, like the Iron fault, having their axes nearly north and south, parallel to the Mosquito range uplift, while the secondary faults of lesser extent, like the Moyer fault, were generally at right angles to the former, striking east and west.

Through these movements the several formations of Iron Hill were tilted to the east and southeast. They dip at an angle of about 14° for a considerable distance, but invariably flatten out and take a basin-form upon approaching the next succeeding fault, or, where the elastic limit of the strata is exceeded, break into minor faults parallel to the general direction of fracture. These step-faults or jointing-planes, as shown in Section AA′, on the westerly extension of the blue limestone below the Iron fault, gradually decrease in intensity and disappear in the basin-like or synclinal form of the limestone below, and probably represent the settling or concluding dynamic action of the region. They are visible both in the ore-bodies, and in the enclosing igneous and sedimentary rocks.

Through the effect of subsequent periods of erosion and denudation, the present surface contour of Iron Hill has been formed. The

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upper sedimentary formation of Weber shales and grits has been entirely removed. The white porphyry has been left as the surface or country rock, and the thicknesses of the several geological formations are about as follows:

Archæan granite, unknown.

Cambrian quartzite, 150 feet.

Silurian white limestone, 160 feet.

Igneous gray porphyry, 105 feet (including the parting quartzite of Emmons).

Carboniferous blue limestone, 225 feet.

Igneous overlying white porphyry, from nothing to 600 feet.

In this locality no developments have been made that show the existence of the parting quartzite of Mr. Emmons. Its thickness is therefore included in that of the gray porphyry, of which formation it appears, for reasons to be given further on, to form a part.

Faults.

Considering Iron Hill as an irregularly shaped block of easterly and southeasterly dipping beds of sedimentary formation, enclosing intruded porphyries, it is bounded on four sides by four principal faults: On the west, the Iron fault, with a strike nearly north and south, and a throw decreasing from its southern to its northerly extension; on the east, the Mike fault, parallel in strike to the Iron fault; on the northern slope, the Adelaide fault, dividing it from Yankee Hill, and on the southern slope, the Moyer and California fault, dividing it from Rock Hill. The latter three have the general east and west direction, the two southern faults running parallel to California gulch, and increasing in throw from their junction in an easterly and westerly direction, the Moyer fault having a throw of several hundred feet at its connection with the Mike fault.

Dykes.

As will be readily seen from an examination of the accompanying sections through Iron Hill, the intrusions of gray porphyry have been much more general in its eastern and southeastern portions than elsewhere. Parallel to the present course of California gulch, and across the claims of the A. Y. and Minnie, Col. Sellers, and Moyer Placer, the largest intrusions of gray porphyry have apparently taken place. But the gray porphyry intrusions rarely show the planes of movement, due to the final action of subsidence and con-

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traction of the sedimentary rocks, causing minor faulting, as distinctly as does the limestone, or even the older white porphyry. For in the case of the thin tongues and sheets, the gray porphyry, being still in a semi-plastic state, under heat and great pressure, was only bent and dislocated, while the colder and more rigid strata were cracked and broken.

Near and on both sides of the plane of the Moyer fault, the gray porphyry intrusions have been forced in irregular tongues and dykes through the blue limestone, and up to the contact of the white porphyry above, where they flattened out into sheets of considerable extent and thickness, directly underlying the white porphyry and forming the hanging- or top-wall of the ore-bodies in the mines of this locality. At a lower horizon they split between the bedding-planes of limestone, in the form of thin intercalated sheets of great horizontal extent. North of the Moyer dyke, another dyke of size and importance is disclosed in the North Imes claim. This dyke, cutting nearly normal to the strata of the limestone, has a northeasterly course, and is disclosed at different levels in the workings of the Imes and Colorado No. 2, and is probably the same as that cut in the Horse Shoe and Newton-Ulster shafts. From this dyke, near the middle and lower third of the blue limestone, a thin sheet of gray porphyry of great extent has forced itself between the bedding-planes. This sheet, having an average thickness of 25 feet, has been cut over an area extending from the Horse Shoe shaft to the eleventh-level winze of the Iron south incline, along the Ruby level, West Ruby level, and bottom of the Cord main incline, and in every development which has penetrated the limestone to that depth. In the Cord incline it is encountered at its proper level as deduced from the points of contact with it in the other workings of the Cord, Louisville, and Iron mines, and is traced directly to its source from the Imes dyke. From this sheet or dyke, other cross-dykes are again intruded, as shown in the Colorado No. 2, eighth level of the Iron, Willard level of the Cord, and numerous other instances.

Other flat sheets of importance are found in the north drifts of the A. Y. and Minnie claims and bottom level of the Cord, as shown in Section AA′. This porphyry sheet, having an average thickness of 20 feet, directly underlies the White Cap and Smuggler ore-chutes, and has been clearly defined, as shown in the workings of the Cord, Minnie, and Sellers mines. In the northern portion of Iron Hill, the most extensive gray porphyry intrusions, so far as developed,

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have been found in the porphyry winze of the Iron north chute and in the McKeon sixth level, over which it extends. It is also said that the large and unusually rich bonanzas of the Mikado mine, lying west of the Iron fault, are directly associated with several flat sheets of gray porphyry, extending from the top to the bottom of the blue limestone, in the form of successive contact-deposits. In considering the numerous occurrences of the gray porphyry, which split and cross-cut the limestone at every angle and horizon, it is difficult, in the study of so limited a scope of country, to define their principal source or their connection and relation to one another. It is probable that they are all offshoots from the general intrusion between the Carboniferous and Silurian limestone.

It has been observed, however, that the larger dykes across these sedimentary strata conform, in most cases, to the same general system, following a northeasterly and easterly course; a system in which dykes, ore-chutes, and faults have been intimately associated.

Contact Metamorphism.

Where the several dykes and offshoots of gray porphyry have been studied, the metamorphism of the rock in contact with them shows a most remarkable absence of heat alteration, as has been previously stated by Mr. Emmons, on page 307 of his report above referred to, the only change observed being that on approaching a dyke, the limestone, a few feet distant from its flank, is harder, more siliceous, and finer grained. No instance of marbleization has been seen in the limestone in contact with the porphyry, and such negative evidence tends to the conclusion that the slight alteration observed has been due rather to great pressure and subsequent water-action, than to any intense degree of heat. On the other hand the porphyry shows a more marked alteration. In vertical dykes cutting across the strata, there is no silicification, but the porphyry is decomposed and almost kaolinized for several inches from its contact, or, when in the form of thin vertical tongues, completely altered for its entire thickness; if intruded at a low angle for any distance between the strata of sedimentary rocks, as with the thin sheets of porphyry from the Imes dyke, the reverse condition of contact phenomena is invariably noticed. In this condition the porphyry sheets are altered almost to a normal quartzite, without any sensible change in the enclosing limestone, except that a black, shaly selvage and an occasional thin zone of dolomitic lime-sand are found, indicative of pressure and recent decomposition by water. In such cases the structural and crystal-

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line characteristics of the porphyry entirely disappear, and give place to the white color, and sub-vitreous, close-grained texture of quartzite. This metamorphism extends from 1 to 10 feet from its outer contact, and in thin sheets entirely through them. Analyses of this metamorphosed porphyry show from 80 to 89 per cent. of silica, decreasing to the normal percentage of about 67 per cent. of silica, where merging imperceptibly into the unaltered portions of the main dyke. In the case of dykes cutting across the strata, the metamorphism may be attributed to the action of heat and water, the percolation of waters through their natural channel (the jointing planes of limestone) being here interrupted by the flanks of the dyke itself, which present a greater area to their decomposing action. In the case of flat sheets between these bedding planes, comparatively small opportunity has been offered for the disintegrating and decomposing action of the water, and through the combined influence of heat and pressure, together with a probable concentration or robbing of the silica from the small portion of decomposed lime above and below, the metamorphic alteration to an apparent quartzite has been induced. This same condition of the contact of the gray porphyry and limestone is observed at the lower horizon of the blue limestone, both in the McKeon shaft, which only penetrated it near the Iron fault, and in the lower workings of the Louisville mine and Colorado No. 2 shaft, where it has been followed horizontally for a distance of 800 feet. In the latter instance, as shown in Section B B′, the Louisville workings, at a depth of 190 feet in the upper limestone, have cut through, or nearly through, the entire thickness of the blue limestone, and at this depth encountered what was considered to be the parting quartzite, and probably is the parting quartzite of Mr. Emmons, though in reality it is a metamorphosed porphyry. In this position, this quartzite, or metamorphosed porphyry, with an average thickness of 20 feet, merges, without any line of bedding, contact, or demarcation, into the unmistakable gray porphyry below, in which the bottom of the Colorado No. 2 shaft, 40 feet deeper, now rests. In its upper contact with the limestone, it is very irregular, and intruded in thin tongues, which are largely decomposed and unconformable with the limestone, while in the mass of the quartzite layer unaltered gray porphyry can be found. Mr. Emmons's report on the Leadville district states that the parting quartzite of Iron Hill is not positively determined. Nor can it be satisfactorily seen in any of its extensive mine-workings. On Carbonate Hill, as on Iron Hill, the blue limestone is found split at different horizons by the intruded metamor-

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phosed sheets of gray porphyry, under and over which lower ore-horizons have been developed, giving rise to the "second" and "third contact" theories of this hill. The "persistent" bed of parting quartzite, however, is not found.

In other localities, Mr. Emmons has generally placed the parting quartzite either at the upper or the lower contact of the gray porphyry with the two limestones. In the numerous workings on Iron Hill, both of these contacts have been opened without disclosing it, and in the same Carboniferous and Silurian limestones of Battle Mountain, near Red Cliff, where the gray porphyry is wanting, the parting quartzite is likewise absent. While admitting the common occurrence of sheets of quartzose rock with the gray porphyry, between the Carboniferous and Silurian formations, it has been included in this paper in the thickness of the gray porphyry, and is believed to be an irregular zone of contact alteration, or metamorphic quartz-porphyry, rather than quartzite of Silurian formation; the parting quartzite of Iron Hill, at least, having no existence.

Relation of Ore-Chutes and Ore-Channels to the Dykes.

In the extensive underground work which has been done in the Iron Hill mines during the past ten years, the development of the gray porphyry dykes has naturally advanced more slowly than that of the ore-chutes and channels. At first, these foreign and unexpected formations, when encountered, were entirely disregarded by the miner as having no important bearing upon the ore-bodies, and were designated by all manner of erroneous terms, such as quartz seams, horses, white lime, and tongues of white porphyry. With the thinning out of the contact and carbonate ores, however, and the opening of ore-bodies of deeper horizon and of the present sulphide ores, the development of the gray porphyry dykes and sheets, and their intimate relation and association with the ore-chutes, have become by experience and general acceptance points of importance in prospecting for ore-bodies only second to that of the blue limestone, the one being apparently but the receptacle for the resultant action of the other. On Iron Hill developments of this kind have been greater than in all the other mines of the Leadville district combined, and nowhere else are the above facts so clearly and beautifully exemplified as in the eastern workings of the Iron, Cord, Minnie, Louisville, Moyer, and Sellers mines.

As before stated, in passing from the northwestern to the south-

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eastern portion of Iron Hill, from a region of lesser to one of greater disturbance and crushing, the ore-deposits within the several chutes occur more irregularly and deeper in the body of the blue limestone than in the well-defined contact-chutes; in which respect the flattened North Iron chute may be compared, in contra-distinction, to the Cord and Minnie chutes. It is therefore within the workings of these latter mines that the relation of the gray porphyry to the ore-bodies is most clearly exhibited. In the sectional maps, which are accurately drawn to scale, from a study of the mine-workings, these facts are more comprehensively presented than could be done in any other way, though absolute accuracy of detail cannot in all instances be claimed for them, a certain geological license being unavoidable in projecting known relative features of formation through regions of undeveloped ground. In all these sections, however, the relation of ore-body to gray porphyry is apparent. In the northwestern portion of Iron Hill the porphyry intrusions have not been as extensive, nor have they reached the same horizon in the limestone, as in the southern and eastern portion. It is probable, however, that deeper developments in the properties of the Iron mine, Smuggler, and Iron Hill Consolidated Mining companies, dykes and flat sheets of gray porphyry will be disclosed, which have directly influenced the deposition of the ore-chutes and channels now shown to extend across them, but undeveloped in their lower horizon. In the Gold-ore chute of the Silver Cord mine, as shown in Section AA′, which has been worked to a greater depth in the limestone than any other in the district, a porphyry dyke of low horizon has been cut in several places on its northwestern flank, though developments have been insufficient to follow definitely and to locate its direction as related to the chute itself; while in the deeper workings of this mine, the Imes, North A. Y. or White Cap, and Smuggler chutes are found directly following and parallel to the trend of the Imes and other dykes. Also, in the case of the Stone chute, the zone of replacement and ore-deposition directly follows the Moyer dyke, and being the largest ore-chute yet disclosed in the entire Leadville district, it is not only more closely associated with it, but apparently its mineral contents are in proportion to the great volume of the gray porphyry enclosing and adjacent to it. In general, however, the ore-channels do not always come in actual contact with the dykes, but follow the zone of replacement of the lime at a considerable distance from them; and where such contact is observed, the ore is lean and poor in silver and lead, composed mostly of decomposed friable

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iron pyrites with numerous quartz crystals, but showing appreciable quantities of gold.

In Section BB′ through the Cord incline this condition is clearly shown. The Imes, White Cap, and Smuggler chutes are parallel to the Imes and Ottawa dykes, with a second dyke between them, and in the case of the White Cap chute, the white porphyry above the ore-body, as well as the chute and gray porphyry below, shows distinctly the minor faulting previously alluded to.

As a guide to future developments on Iron Hill, it may here be well noted that wherever a dyke of considerable extent has been heretofore disclosed, an ore-body of proportionate size has been encountered following it.

It moreover appears from the developments made to date, that especially in ore-bodies below the white porphyry contact, the form of such deposits assumes in general the form of its correlated gray porphyry intrusion. For example, the second and third contacts of Carbonate Hill lie above and below flat sheets of gray porphyry. In the extension of the Cord Gold-ore chute the ore-body changes, with a sudden change of horizon, from a vertical to a horizontal position and flattens out on two flat sheets of gray porphyry, though often separated from them by sheets of hard, siliceous, unreplaced lime. The same principle is observed in the Stone and Smuggler chutes, while in the Ruby, A. Y., Cord, and White Cap mines the vertical channel-form of deposit follows the vertical porphyry dykes.

In the plan-map accompanying this paper, all the surveyed and platted underground workings of this portion of the hill, up to the date of its completion, are shown. Through the courtesy of the managers and owners of the several mining properties, the writer has had, in addition to unreserved access for examination of their mines, and full information regarding them, the unusual courtesy of copies of their underground maps, of which the accompanying map is an accurate pantagraph reduction and compilation.

Ore-Chutes.

From the upper contact with the white porphyry to the greatest developed depths, and presumably to the very lowest strata, the substitution and replacement of the Carboniferous limestone by ore and vein-material, has taken place irregularly in parallel zones, having a general northeasterly direction. Within these zones of replacement the bodies of ore and vein-material of Iron Hill occur in continuous ore-chutes or channels occupying different horizons

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and undergoing small local changes of direction, but uniformly returning to their northeasterly trend. In a general sense, it is difficult to draw any line of marked distinction between the ore-chutes occurring in shallow troughs at the upper horizon of the limestone, as in the large Iron-mine contact-chute, and those formed deeper, as with the Cord Gold-ore chute and others south of it.

In their essential features, both the upper contact-chutes and the deeper limestone channels are the same, both having been formed from the same sources, only at different horizons, and following the same uniform direction, the one being often traceable into the other through numerous cracks and ore crevices, formerly not considered worth prospecting.

As of next importance to the now well-established principles of direction and continuity, the characteristics of the ore-chutes at different horizons may be noted. It is first observable that in the portions of the sedimentary beds which have been comparatively undisturbed by dykes, the principal replacements of the limestone have occurred on the upper contact, the resultant ore-deposition being formed generally in flat, shallow deposits from 50 to 250 feet in width, and from 1 to 20 feet in depth.

In the deeper channel replacements of the limestone, the ore-bodies are much more irregularly distributed, occurring in lenticular masses and chambers, which are always connected with each other and frequently with the contact above by contractions and pinches of the chute or channel. This latter type of formation is invariably in the area of greater disturbance, and is generally found following a porphyry intrusion of corresponding form.

In comparing the vein-material, the upper contact-deposits, where altered to a carbonate or oxidized ore, are generally more highly acid or siliceous, while those in the body of the limestone are more basic and composed largely of the oxides of iron and manganese carrying a greater value in silver. The occurrence of gold (if found at all in the chute) is also more common near the contact of the gray or white porphyry. The ore-chutes on Iron Hill, as indicated upon the plan-map by shaded lines, are severally designated as follows: The North Iron chute, the South Iron chute, the Cord Gold-ore chute, the Ruby Channel chute, the Imes chute, the White Cap chute, the Smuggler chute, and the Stone chute.

These chutes are named from the properties in which they were first discovered, or have been most extensively developed. From four of them, viz.: The North Iron chute, South Iron chute, Gold-

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ore and Stone chute, fully 90 per cent. of the entire ore-product of Iron Hill has been obtained. The remaining four chutes, owing to their having been only periodically worked and unsystematically developed, in most cases by lessees, are not so continuously defined as the former. Through the several connecting links, however, of workings and stopes, their size, form, and direction, as well as the characteristics of their ore-bodies, can be clearly and accurately traced and produced.

The Iron Chute.

This ore-chute may be regarded, so far as it has been developed, as a type of the first discovered contact-deposits of the Leadville district. On Iron Hill it has in the past been only second in size and value to the Stone chute in California gulch, as shown by the broad and continued stope on the map. It has been worked at enormous profit by the Iron Silver Mining Company, from its apex, at its junction with the Iron Hill fault, for a distance easterly of 1500 feet, and westerly of 1900 feet. It occurs, so far as developed, as one nearly homogeneous body of carbonate-ore, lying immediately under the white porphyry, and filling the shallow basin-like replacement of the blue limestone to an average depth of 8 feet below the contact. Its width is about 350 feet, pinching up to tight contact on its flanks. Its trend is N. 40° E., following the upper surface of the limestone at a dip of about 12°. In its extreme eastern continuation the North Iron chute has apparently pinched out, and become lost in the lime-porphyry contact. This may be considered, however, only as a most remote possibility, as subsequent developments will show the chute reappearing after passing the pinch, or, more probably, that it has pitched down into the limestone and chosen a different horizon, as shown in the Louisville and Colorado No. 2 extension of the Gold-ore Cord chute, Section BB′. In numerous places in this chute, notably at the eighth level and in the porphyry winze, which is sunk 200 feet in a porphyry dike, offshoots and stringers of ore have been found extending down to varying depths into the body of the limestone, as shown in Sections BB′ and DD′. Until quite recently no systematic developments of these bodies of ore, upon the gray porphyry, have been undertaken by the Iron Company, though it is but a fair and safe presumption that the developments now in progress, in the McKeon sixth level east and eleventh-level winze, will result in opening large ore-bodies beneath the contact, and associated with the intruded gray porphyry

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sheets and dykes shown in Sections DD′, AA′, and BB′. Westerly from the top or apex of the blue limestone, the North Iron chute, having been broken or faulted with the entire hill formation, is found again occupying the same geological horizon, and with a slight lateral movement to the north, but retaining the same north-easterly and southwesterly course. The Iron Hill fault, together with the westerly continuation of this chute, has been here thoroughly developed by the McKeon inclined shaft, which is sunk at an angle of 50°, just east of the fault-plane and near the basset edges of the limestone, as shown in Section DD′.* By levels driven westerly from this shaft, the fault is found to have a throw of about 500 feet, with an angle to the westward of 60°, or nearly perpendicular to the eastern dip of the upper strata. West of the fault the limestone is found dipping westerly away from the fault and breaking into consecutive minor faults, in the form of steps, which decrease and disappear in the bending and basining of the limestone, and continue west into the easterly dipping beds of Carbonate Hill. The westward continuity and characteristic features of the ore-chute have been fully demonstrated by the workings on the Star of the West, the Blind Tom, and Satellite mines, the only difference being that the ore-bodies are always found best upon the surface of the limestone (seldom, if ever, upon its basset edges), and within the plane of the minor fault; and in general with a decreased thickness, which is more than compensated for, in a financial sense, by the increase in grade of the ore. No gold, in paying quantities, has ever been found in this chute. Its product, so far, has consisted entirely of carbonate ores, though sulphides will be found in its easterly continuation.

The South Iron Chute.

The apex of this chute is found in the zone of the blue limestone, lying between the No. 1 and No. 2 inclines of the Silver Cord mine. Owing to insufficient developments it is not as clearly defined at this point as in others. By levels driven south from No. 1 incline, it is found near the porphyry contact in numerous unexplored crevices and channels, extending down into the limestone. North of No. 1 incline, and at a greater depth below the contact, it has been more

* A portion of Section DD′, through the McKeon shaft and west of the Iron fault, was constructed by Mr. Noble, engineer of the Iron Silver Mining Company, who kindly permitted its use in this paper.

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extensively developed and worked. At this point it exists at a depth of 50 feet in the limestone, in the form of large bodies of oxides of manganese and iron, carrying silver, but no lead. In its eastward continuation, after passing the Silver Cord shaft, it has apparently divided into two branches, which, with a slight convergence, after passing the line of the Iron mine, meet the contact on its downward pitch and immediately spread out in the form of a flat contact deposit, having an average thickness of 6 feet and a width of 220 feet. In this form it continues obliquely across the tenth and eleventh levels of the Iron south incline, to within about 100 feet of the Louisville shaft, where, after sending out cross-chutes or branches to the other chutes north and south of it, it apparently pinches out like the North Iron chute, but has been found again in its normal trend, on a lower horizon, following a second contact of gray porphyry. Near the center of this chute, between the tenth and eleventh levels of the Iron south incline, as shown in Section EE, the eleventh winze has been sunk. This winze, after passing through 90 feet of limestone and 30 feet of gray porphyry, has encountered sulphide ore of good grade, and is one of the most important developments recently made on Iron Hill. Work in this ore-body, which has been previously prevented by water, has been recently resumed, and will result in important discoveries of ore similar to the second contact-bodies of Carbonate Hill. After passing from the limestone to the contact the character of the ore in this chute changes to a siliceous lead-ore, low in silver, with no gold; while in the eleventh-level winze before mentioned, the ore-body is found mostly composed of sulphides of iron, with but a small percentage of lead and zinc.

The Gold-ore Chute.

This chute, so called on account of the unvarying and persistent presence of gold in all its ores, is, because of its extensive development, the most interesting ore-deposit of the Leadville district. As it probably will not be found to differ widely from other chutes, a careful study of its formation and developments furnishes valuable precedents for prospecting the body of the limestone in the other cases, and with promise of equally successful results. As will be observed on the map, this chute has been continuously developed for a distance of 2500 feet, and is the farthest advanced towards the east of any chute of the region. In section AA′, its levels and horizons are accurately shown.

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The outcrop and apex of the Gold-ore chute are found at the scored surface of the blue limestone on the south end of the Silver Wave and Bull's Eye claims. For the first 300 feet, it occurs in two principal branches, in which the ore-bodies are formed in most irregular but connected chambers or pockets extending from the contact to a depth of 180 feet into the limestone, and probably entirely through it. In its continuation northeast, the contact-ore entirely disappears, and the chute becomes a perfect net-work of crevices and channels of ore and vein-material, standing perpendicular to the strata of the limestone, and often entirely enclosing blocks of partially replaced lime-sand and limestone. After crossing the main Cord incline, one strong cross-channel passes off northwest to a junction with the North Iron chute, and then, consolidating into two convergent channels, it continues in a nearly horizontal position, until at the Louisville west side-line, it reaches the porphyry contact. At this point, it undergoes exactly the same changes observed in the two Iron chutes just described, reversing its former position from a nearly vertical to a horizontal one, and spreading out into a large contact ore-body. In its further extension east it pinches out in the lime-porphyry contact, after sending out northwest and southeast branches and clasping hands, as it were, with the chutes on each side.

As is not the case with any other ore-chute on Iron Hill, however, two, at least, of the small crevices extending down into the limestone from the upper contact have been followed. Through these important developments, the Gold-ore chute was found to make a sudden pitch downward, for a vertical distance of 120 feet, until, meeting the intruded sheet of gray porphyry, as shown in Sections AA′ and EE′, it was temporarily arrested, and followed the second horizon for several hundred feet across the Louisville claim. Near the east side-line of the Louisville, the ore-chute again pinches up, but, maintaining its downward tendency, is traced by small feeders through the gray porphyry, 20 feet in thickness, to a second gray porphyry sheet or horizon, 120 feet below the first. On this, its third horizon, in a horizontal distance of 400 feet, as shown by developments from the bottom level of the Colorado No. 2 shaft, the Gold-ore chute again flattens out, and, following the top of this gray porphyry, but without being always in direct contact with it, continues in its northeasterly course, across the Colorado No. 2 claim, into the east end of the Tucson claim.

In its further easterly continuation it will probably cut the Horse

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Shoe claim about 150 feet north of that shaft and at a still deeper horizon. The easterly portion of Section AA′, through the latter shaft, does not therefore follow in the chute, but cuts obliquely across it. Up to the point of its rising and spreading out under the lime-porphyry contact, and subsequently becoming dispersed and pinched out in its easterly continuation, the Gold-ore chute does not differ essentially from any of the other chutes of Iron Hill, which have been followed and developed up to this line. The value of its advanced developments, showing its downward pitch and continuation on the second and third horizons of intrusive and partly metamorphosed porphyry, cannot be overestimated, as this fact explains the apparent ending of the two Iron chutes, and demonstrates almost beyond a doubt, that their continuation will be found in undiminished strength through similar developments and upon corresponding horizons.

The Ruby Channel Chute.

The Ruby Channel chute is first disclosed at the apex of the Iron Hill contact in the north end of the Lime claim. It is here developed in the Doyle shaft, 60 feet deep, and in the incline therefrom, as a deep limestone deposit, consisting entirely of oxides of manganese and iron, with no lead, but with from 10 to 100 ounces of silver per ton. Its trend is about N. 40° E. From the workings near its outcrop, it extends across the north end of the Smuggler claim, and is next shown in the South Cord level, though no developments have been made in it at that point. After crossing the Smuggler claim, it is again disclosed in the South Ruby level, from which point it has been continuously followed by the Ruby level to the south end of the Louisville claim. After crossing into the latter, it rises, as in the case of the other chutes, to the contact, spreading out and joining the Gold-ore chute on the north, towards which it maintains a convergent course from its outcrop. In the northern half of the Ruby claim, it disappears in the upper contact, but its continuation has been demonstrated deeper in the limestone and beneath the intruded gray porphyry sheet in the lowest levels of the Ruby and Colorado No. 2 shafts. This chute, as shown upon the map, is very uniform in size and direction. Where thoroughly developed in the Ruby level and Doyle workings, it has a width of about 30 feet, with limestone walls clearly defined, and, as shown in Sections BB′ and EE′, extends from the upper contact, with a westerly pitch, to the intruded gray porphyry sheet, at which point

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it pinches out almost entirely. Below this level, deeper developments will probably result in important discoveries.

The Imes Chute.

With the single exception of the uniform presence of gold in all of its ores, this chute is, so far as explored, a nearly parallel reproduction of the Ruby chute just described. As a narrow limestone channel of about 20 or 30 feet in width, it is traced from its apex on the Lime claim, through the Imes claim of the Iron Hill Consolidated Mining Company, until it reaches the contact just after entering the property of the Silver Cord Company. From this point, in its easterly continuation, its ore changes suddenly into a sulphide of low grade in silver and lead, but carrying from 20 to 40 per cent. of zinc and $\frac{2}{10}$ to $\frac{8}{10}$ of an ounce in gold. In its extreme eastern development, near the Willard level of the Cord mine, it is split and dispersed by thin decomposed tongues of gray porphyry, which are off-shoots from the Imes dyke on its western flank, but is clearly seen dipping down and seeking a lower horizon upon which it has not yet been developed. Vertically this chute extends from the white porphyry contact to the gray porphyry sheet, at which point it disappears and pinches up as shown in Section EE′, as above described of the Ruby chute. The greatest gold value occurs invariably nearest the white and gray porphyries.

The White Cap Chute.

The first 700 feet of the White Cap chute has been clearly developed in the workings of the Lime, Smuggler, Forfeit, Ocean Wave, and White Cap claims, across which properties it passes. Across the three latter claims, belonging to the Iron Hill Consolidated Mining Company, it has been but partially developed, and worked as a contact ore-body resting upon low-grade iron-ore and vein material, extending to unexplored depths in the limestone. In its easterly extension, it crosses the north end of the A. Y. claim, and is cut in the A. Y. north level. From this point, following near the centre of the Ottawa claim of the Silver Cord Company, it has, within the past six months, been continuously developed, its most advanced easterly exposure being in the Willard level, and in the Eagle raise from the bottom level of the Cord mine, where it carries sulphide ores.

Sections FF′ and GG′ are accurate representations of this chute, on a scale of 50 feet to the inch. The former is drawn looking

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northeast and across the trend of the chute, from the northeast corner of the Minnie claim; section GG′ is a longitudinal section looking northwest, drawn through its center, parallel to the east line of the Minnie claim. In these two sections many of the characteristic features of the other chutes can be seen. It is shown to lie between and parallel to two porphyry dykes, the larger (Imes) dyke on the west, and a smaller dyke disclosed in the Cord, North Minnie, and Sellers workings on the east. Underlying it is also a flat sheet of gray porphyry, about 25 feet thick. The several minor faults are also shown in the section. About 15 feet northeast of Section FF′ the ore changes suddenly, in a few feet, from a carbonate to a heavy sulphide, carrying an unusually high percentage of zinc and iron, with but little lead.

From the crushed and broken appearance of the limestone beneath these chutes and in the Cord bottom level, it is thought not improbable that a second ore-body will be found under the first and beneath the gray porphyry sheet, corresponding to the second contacts of Carbonate Hill, and as shown on general Section BB′.

The Smuggler Chute.

Though one of the earliest discovered and earliest worked ore-bodies of the Leadville district, the Smuggler chute has in later years received very little attention, and consequently has less longitudinal development than any other well-defined chute on Iron Hill. This unfortunate condition has been caused by a combination of adverse circumstances, arising from the cessation of work, pending the long and tedious apex-cases between the Lime and Smuggler claims, and the subsequent shutting down of the shafts of the Smuggler Mining Company and the Iron Hill Consolidated Mining Company, across whose properties this chute has been disclosed. In one respect the Smuggler chute may be regarded as the northern flank of the great Stone chute, south of it. Its outcrop and apex, however, have been carefully determined at the contact, and within the scored strata of the blue limestone upon the Lime claim. It has, thence, been extensively developed by workings from the old Lime incline across the Smuggler, and into the property of the Iron Hill Consolidated Mining Company. For at least a portion of this distance it is overlain by an intrusive sheet of gray porphyry, a branch or tongue from the Moyer dyke.

After passing the White Cap shaft, it is seen with an increased width in the northern workings of the A. Y. claim, beyond which

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point no developments have been made at the proper horizon to encounter its further continuation. The explorations of this chute, like those of the North Iron chute, have been confined mostly to the upper contact, at which horizon the ore-bodies have been generally found to exist. Until a very recent date, explorations of the numerous fissures or channels extending deeper into the limestone have not been attempted, but this is now being done with successful results by Mr. John F. Campion on the Smuggler claim. In its easterly extension, the north drift of the Sellers, along the Minnie line, is deep in the limestone, and too low to cut it at this point where it is probably on the upper contact. Its extension, however, will pass through the Minnie, Sellers, and Willard claims, as indicated on the plan-map.*

The Stone Chute.

The outcrop of this chute in the bed and on either slope of California gulch was the locus of the first discovery of carbonate ores in the Leadville mining district. In its early history it was exclusively worked in the Stone, Little Daisy, Gilt Edge, and South A. Y. claims, under the then existing belief that the upper contact was the only profitable ore-horizon of the limestone. Through its subsequent enormous development, however, it has been demonstrated to be the zone of largest limestone replacement and richest deposits of ore on Iron Hill, and probably in this entire mining region. The Stone chute has been developed principally north of California gulch, though what might be considered a branch of it may yet be found south of the gulch, and in the continuation of the limestone, which has been split by the ejection of the Moyer dyke. From its most westerly exposure in the Rock, 76, and Stone claims, it has a trend of about N. 35° E., crossing the lowest portion of the A. Y., Minnie, and Col. Sellers properties. Its furthest eastern development is in the workings of the Sierra Nevada claim, from the Col. Sellers Accident shaft, and in the Moyer workings of the Iron Silver Mining Company, and is undoubtedly the same chute disclosed in the Mike and Star mines. In the occurrence of ore-bodies in this chute, the limestone replacements in proximity to the erup-

*Since the above was written, the Smuggler chute has been disclosed at the upper lime-porphyry contact, near the southwest corner of the Willard claim, by two raises from the Minnie and Sellers line drifts, and 55 feet above the latter. It here consists of the characteristic sulphide ores of the region, 12 feet in thickness. The width is not yet determined. Its grade in silver and lead is apparently *about the same as that of the Stone chute.*

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tive rocks have been enormous, resulting in the largest deposits of ore yet discovered in Leadville, being in many cases over 100 feet in thickness. Near the surface in the Stone workings the ore is found as a carbonate, though north of the gulch it consists principally of large bodies of the sulphides of iron, lead, and zinc. Fully 75 per cent. of the entire mass of these sulphides is of a very low grade, carrying from 6 to 15 ounces of silver, and corresponding closely, in their contents of the precious metals, to the product of secondary alteration in the large bodies of iron and low-grade vein-material. Within this mass of low-grade sulphides, however, and indistinguishable from them by the naked eye, are found irregularly shaped bodies and streaks of ore of exceptionally high grade in silver. Neither the sulphides of iron, lead, or zinc seem to have exerted any special influence or affinity for the deposition or re-deposition of the silver. The greatest values of the latter are found in the same ore-body, generally in the form of native silver, combined distinctively with a preponderance of first one and then the other of the former sulphides. A description of the formation and mode of deposition of the ore-bodies in the Stone chute can best be obtained from an examination of Section CC′, and reference to the valuable article of Mr. F. T. Freeland* on "The Sulphide-Deposit of South Iron Hill." It may be confidently asserted, however, that the easterly extension of this ore-chute will eventually embrace one of the largest and most productive portions of the region.

In this brief account of the several ore-chutes it has been the object to furnish some evidence from which valuable deductions can be made as to the future developments of Iron Hill, as well as of property east of it, and other portions of this mining district. As a guarantee of the extensive resources and extended prosperity of the Iron Hill region, and of the western slope of Breece Hill, the indications disclosed by the present developments in these ore-chutes lead, necessarily, to two important conclusions. The first is the continuity of these ore-chutes beyond their most advanced exposures, and secondly, the probable existence of ore-bodies on consecutive and lower horizons in the limestone, underneath those already discovered and worked, and particularly upon the lower contact of the blue limestone with the gray porphyry.

The importance of the first proposition is already being fully appreciated by a few of our enterprising mining men, as is attested

* *Trans.*, xiv., 181.

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by the sinking of the Horse Shoe, Newton-Ulster, Park, Belgian, and Lady Alice shafts. During a constant experience and familiarity of nearly nine years with the ore-bodies of Iron Hill, the writer has seen nothing indicative of the "playing out" or end of any of the several ore-chutes. Of the eight distinct chutes of Iron Hill enumerated above, only one, the North Iron, has been developed west of and below the Iron fault, and the developments there resulted in immense success, and fully established its continuity in that direction.* In their easterly extension it will be observed that if a line be drawn perpendicular to the trend of the ore-chutes, and southwesterly from the southwest corner of the Frenchman claim to the southeast corner of the Sierra Nevada claim, the ore-chutes which approach this line, so far as developed, have in every instance met and spread out under the contact, and beyond this point pinched out on this horizon. Beyond this line the Gold-ore Cord chute only has been developed. These pioneer developments, made through the energy and enterprise of the managers and owners of the Louisville and Colorado No. 2 properties, have demonstrated the absolute continuation of the Gold-ore chute at the very bottom of the limestone, and 190 feet below its former horizon. The importance of this discovery to Iron Hill properties cannot be over-estimated. It is most probable, therefore, that the continuation of all the other Iron Hill chutes will be eventually discovered under like conditions, and their extension into the properties in Adelaide Park and Breece Hill is scarcely susceptible of a doubt.

Of the second conclusion, evidences are fully shown in every developed ore-body on Iron Hill. Important developments are now being prosecuted that will demonstrate this; the most interesting being those of the Iron Silver Mining Company, in a level driving east from the sixth level of the McKeon shaft, cutting from the lower to the upper strata of the blue limestone; the West Ruby level of the Cord, cutting through the limestone in the opposite direction, or from east to west; in the sulphide ore of the eleventh-level winze; and through the porphyry winze of the Iron mine before mentioned; also in the fifth level of the No. 2 shaft of the Col. Sellers, and in the lower easterly level of the Silver Cord mine, driven to encounter the four southern ore-chutes of Iron Hill at a greater depth in the limestone.

* The Mikado mine is also located on the westerly extension of the old Adelaide-Argentine chute, west of the Iron fault; but little is generally known of its formation, as its management does not desire to make such information public at present.

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Ore-Deposits.

Within the several ore-chutes just described, and limited vertically only by the thickness of the blue limestone, the ores of Iron Hill are found. They do not differ essentially from other characteristic ores of the Leadville district, especially those in the oxidized or carbonated state, of which numerous complete descriptions and analyses have already been written, but in their widely varying component parts embrace the distinctive ores of nearly all the other portions of the ore-bearing Carboniferous limestone formation.

The ore-chutes contain two distinctive characters of ore, which were originally one and the same: The carbonate or oxidized ores, and the sulphide or unoxidized ores, the latter, though in a more massive and apparently unchanged condition, still showing evidences of slight secondary alteration. In their mode of occurrence in the limestone, and their relation to the intrusive porphyries, the carbonates and sulphides are closely analogous. In form they may be further distinguished as the flat or contact-deposit, observed in areas of least disturbance by porphyry intrusion, and the deeper limestone deposits, or channels most common in the proximity to dykes of high angle. With a change of physical conditions, also, these two forms change one into another, or suddenly seek different horizons, though preserving their uniform northeasterly direction.

Carbonates.

Within the limits of surface-waters and atmospheric agencies, or where found as carbonates, the chutes consist in their major part of hydrated oxides of iron and manganese as a gangue or vein-filling, with clay, silica, and dolomitic lime-sand, the products of the decomposition of the porphyry and limestone. The ores are principally carbonates, sulphates, and sulphides of lead, carrying silver and gold.

The silver is mostly in the form of chloride or bromo-chloride, but occasionally is found as native silver, or with the galena nodules as sulphide. Gold occurs probably in alloy with the silver, and, although found in some ores in considerable quantities, has never been seen in a native state on Iron Hill. It is scarcely possible that it could exist as a chloride, and it has never been found as a telluride. It is persistently and distinctively found in some chutes, and absent in others. Both gold and silver are generally in inverse proportion to the lead contents. Copper has been found in small quantities as a carbonate and native. Zinc near the surface occurs

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only in mere traces, but it is found in larger quantity as depth increases, in the form of a silicate or calamine. Baryta or heavy spar is rarely found. Sulphur is found in the native state, but in small quantities.

A considerable portion of the deeper carbonate deposits are found to consist of basic ferric sulphates, containing sulphates of lead, and similar to the specimens from the Morning and Evening Star mines analyzed by Dr. L. D. Ricketts. This yellow ochreous ore, in gold-bearing chutes, has been invariably observed to be the richest in gold. Other and rare metals have been found, but in such small quantities as to be of no general or economic importance.

In the occurrence and peculiarities of deposit of the carbonate ores within the chute, it is observed that with the flat contact-bodies the ores are more compact and homogeneous. Those lying next the white porphyry (except when near the surface and heavily impregnated with the oxides of manganese) are most heavy in lead, and at the same time have an excess of silica over iron and manganese.

Overlying the heavy lead-ores, layers of clay are often observed, resulting from the decomposition of the porphyry, and stained with the oxides of manganese and iron, but carrying no silver, lead, or gold. Beneath the lead-ore the chute becomes more ferruginous, carrying also more silver. In the bottom, the ore rests upon the limestone without any clay lining, or line of contact, as with the porphyry above, but penetrating it in pot-holes and seams in every direction. These small seams are often formed of galena ore, and are always richer than the lead sand, and cerussite ore above.

In the deeper or channel-form of deposit of the carbonates, the ores and vein-filling are more mixed and irregular, consisting more largely of the oxides of iron than of manganese, with larger masses of clay. The vein-matter or clay often encloses large unreplaced boulders of limestone and lime-sand. In form, these chutes alternately pinch up and open out, both longitudinally and vertically, varying from narrow seams a few inches in width, to large, and suddenly opened chambers or bonanzas. As the chute pinches, the ore always increases in grade and *vice versa*. The cross-sections of these chutes show that they often resemble fissure-veins of only local extent in the blue limestone. Their walls, however, though in many cases very uniform, never show the selvage or lining common to true fissures; and the ore is in many cases attached to, and forms a part of, the wall itself, which it penetrates in cracks and smaller seams. In the Gold-ore chute the most marked example of this phe-

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nomenon can be found; the chute breaking and splitting into a perfect network of small channels, which unite after crossing the Cord level into two, and finally into one, main branch.

When in form of channel-deposit, the chutes generally connect with the white porphyry above, but do not spread out beneath its contact. Their ores are generally more basic and of higher grade in silver than the flat contact-bodies.

Sulphides.

On Iron Hill, as is also commonly the case elsewhere, the ores of the several chutes are richer near the surface and outcrop of the vein and the enclosing blue limestone. Near the surface, also, and nearer the white porphyry, the chutes invariably contain, as a constituent part of the vein filling, larger amounts of the hydrous oxides of manganese, which, it is important to notice, almost entirely disappear upon reaching the limits of the zone of oxidization.

In the northeasterly extension and dip of the chutes into the hill, they all, at or near a line previously referred to, meet the white porphyry contact and, spreading out to the northwest and southeast, connect in a flat cross-chute. Just east of this line, though manifestly through no association with it, they reach the practical limit of surface and atmospheric agencies, and change suddenly, within a few feet, into a massive, heavy, ore consisting of the sulphides of zinc, lead, and iron.

On Iron Hill the change from carbonates to sulphides occurs so rapidly that the miner is suddenly surprised by finding himself out of a body of fine oxidized smelting-ore, and in a close-grained refractory sulphide, consisting principally of zinc and iron sulphides, with sulphide of lead in small quantities. By close observation, however, this point of transition of the ore-chutes can be accurately predicted. With the ore-chutes under consideration these indications are identical with those in numerous other instances observed in other mining districts. It is first noted that the carbonates become more "patchy" or occur in vugs and caves. Among the iron oxides, sulphide of iron next makes its appearance in similar pockets and vugs, and open caves are of frequent occurrence. The iron pyrites first appears in a soft friable ore in a semi-oxidized condition, or generally as a clean pyrite sand with quartz crystals, which is always found in the bottom of the chute under heavy lead carbonates. The carbonate ore, though showing no change which can be detected by the eye, is here found to contain from 2 to 8 per cent. of zinc, but with no increase of sulphur, the zinc occurring mostly as a silicate.

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In the immediate vicinity of the sulphides, the ores are distributed in a loose porous mass of iron oxides, in which open caves are invariably found. These caves, which are caused partly by the shrinking of the original ore-body in oxidation, and more largely by the removal and partial re-deposition of the zinc, are surrounded on all sides by cerussite crystals of different sizes and varying in color from perfectly white to nearly black, the latter being colored by oxides of iron. In the bottom of the caves heavy lead-ores are found, and, underlying them, the pyritiferous sand, resembling carefully sized concentrates, occurs and can be spaded out like sea sand. (See Sections FF′ and GG′). Beyond these regions of change and alteration in physical condition, the ore-chutes pass suddenly, as before stated, into the unaltered sulphides.

Of the eight Iron Hill chutes, all except the North Iron contact chute have now been developed in this character of ore. It can therefore be readily perceived that upon these refractory ores the future prosperity of Iron Hill mainly depends. The sulphide ores consist almost entirely of the sulphides of iron, lead, and zinc, with a small proportion of silica and appreciable quantities of antimony and arsenic. The manganese found so largely in their secondary products, or in the oxidized ores, is only found in inappreciable quantities in the sulphides.

The specific gravity of the sulphide ore is about 3.5, equivalent to 9 cubic feet to the ton, while that of the carbonates averages about 2.5 or 12.8 cubic feet to the ton. These estimates are on the ore as unbroken in the vein and include moisture, of which the sulphides hold about 1.5 per cent. and the carbonates 12 per cent.

In any comparison of the sulphide with the carbonate portions of the chute the general characteristics are necessarily the same; but in the former the metallic contents are distributed in more irregular and varying proportions and the precious metals seem less widely distributed and disseminated through the ore-mass. It is also observed as in the carbonates that the heaviest lead sulphides often, though not always, are found highest up or within the white porphyry contact, while the sulphides of iron occupy lower horizons.

The zinc sulphides are the most widely disseminated, and show plainly the result of their more ready solubility than the other sulphides, and the re-deposition of a large portion of the zinc which has thus been removed from the carbonate ores. This fact is clearly shown in many ways, but most satisfactorily just at the line of transition. The sulphides first encountered are invariably heavy sulphides

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of zinc, carrying a little iron and very little lead. They have a close crystalline structure and lie in a laminated form, the lines of fracture being nearly vertical. Upon these cleavage planes, crystals of cerussite are found, and often a small incrustation of native silver. Such deposits, where first encountered in passing from oxidized to unoxidized ores, are always lowest in silver. In their further extension the zinc gradually grows less and the laminated structure disappears. Beyond this, again, the zinc sulphides appear to predominate along cleavage and contact planes with the gray porphyry, or along the lines of minor faults and cracks in the limestone. Such characteristics are also universally observed in other instances besides those of Iron Hill. In the Stone chute the heavy carbonate ore-bodies of the Stone mine changed first into the iron and then the heavy zinc sulphides of the southwest portion of the A. Y. claim, which are still too low and refractory to extract. In the Gold-ore Cord chute, the Louisville presents the same condition of affairs. In fact, this is a common feature with all the chutes of Iron Hill.

On Carbonate Hill the properties of the Adams and Agassiz Mining Companies have had the same experience, as also have the Aftermath and Milo mines of Ten Mile, and the Belden and Iron Mask of Red Cliff.

In advancing further within the ore-chutes, the zinc appears to lose its preponderance over the other sulphides. This, though not proven, is indicated by the fact that for an average of the entire contents of the chute, the A. Y. and Minnie claims contain more zinc than the Sellers, while with the Moyer, Mike, and Star mines, iron pyrites becomes more and zinc less extensive. In the only other instance of advanced development, the Colorado No. 2 mine, the same thing is disclosed, the ore here being principally iron pyrites and silica, carrying gold and silver. It seems probable that a large proportion of the zinc, which was totally removed from the carbonate ores, has been re-deposited as a sulphide, and principally just below the line of complete oxidation, by surface-waters, and such re-deposition has advanced and increased *pari passu* with the limit and extent of such oxidizing action.

As a corollary of the above, it is believed that at the present stage of development in Leadville, the sulphide of zinc forms a larger part of the unoxidized ores than will be found in future and deeper exploration.

There is also evidence going to show that the action of alteration and secondary deposition has extended for a considerable dis-

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tance within the sulphide bodies. In ores of this class the silver values are found concentrated, as it were, with preponderance of either the zinc, lead, or iron occurring in lenticular masses or patches, surrounded by low-grade ore, and forming bonanzas of great value. These bonanzas are rarely found near the flanks or sides of the chute, but generally in its center and with no connection one with another. Their silver contents consist most largely of native silver, as accumulations of wire-silver with quartz crystals, filling small cavities and vugs in the ore-mass, or as incrustations or foliated patches of the native metal deposited upon cleavage planes and joints. The wire-silver is most common in the lead or galena ores, whilst the native metal, in the form of fine incrustations, is characteristic of the zinc sulphides.

Caves of any considerable size are never found in the sulphide ores, though openings are occasionally seen in small water channels filled with iron oxides, and there are occasionally small pockets of semi-carbonated ores.

An exact classification of the sulphide ores of Iron Hill, with regard to their relative contents of the precious metals and consequent values, would be impossible. It appears to be evident, however, that the gold is associated with the iron pyrites, as shown in the Moyer, Colorado No. 2, and Cord mines.

The following incomplete analysis of some of the Iron Hill sulphide ores will give a very correct idea of their principal component parts. The ores analyzed were taken from an average of from 10 to 25 lots shipped to the smelter:

	Col. Sellers Mine. Average of 25 Lots, 2000 tons, Lead Ore. Per cent.	Col. Sellers Mine. Average of 12 Lots, 1000 tons, Zinc Ore. Per cent.	Colorado No. 2 Mine. Pyritiferous Gold-ores.
Pb	27.40	10.70	.....
Zn	25.00	24.50	.....
SiO_2	3.00	3.40	53.50
Fe	6.00	16.60	20.00
S	35.00	40.00	21.50
H_2O	2.00	2.10	.....
Au	Trace.	Trace.	0.25 oz. per ton.
Ag	26.9 oz. per ton.	54.3 oz. per ton.	55.0 oz. per ton.

Origin of the Ore-Deposits.

In the preceding portions of this paper the purpose has been to describe as briefly as possible the structural features of Iron Hill, together with some of the most important conditions of its ore-de-

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posits. In addition to the occurrence of the ores in well-defined ore-chutes, their continuity and relations to the adjoining eruptive and sedimentary formations, which are equally applicable to other portions of the Leadville district, the extensive and systematic developments of Iron Hill mines offer unusual advantages for a theoretical study of the probable origin or source of the ore-deposits.

As a recapitulation of the foregoing, the facts regarding the Iron Hill ore-bodies may be summarized as follows:

First. The ore-deposits occur only (so far as known) in the blue Carboniferous limestone, occupying irregular portions of the entire thickness of this formation.

Second. The deposits, though undergoing vertical changes of horizon, preserve a uniform northeasterly trend or direction, forming continuous ore-chutes, which were formed subsequent to the porphyry intrusions, and prior to the dynamic movements of the region.

Third. The ore-chutes show no decided relationship to the overlying white porphyry, but, on the contrary, conform in a remarkable degree to the later intrusions of gray or dyke porphyry, which latter has unmistakably influenced their deposition.

Fourth. The disturbance of the sedimentary rocks, incident to the gray porphyry intrusions, caused a crushing and buckling of the limestone strata, which prepared them for the more ready circulation of the mineral-bearing waters along their contact planes and within such zones of disturbance, and the consequent substitution and replacement of such crushed and broken limestone in parallel chutes following the general system of the gray porphyry intrusions.

Fifth. In regard to the ores it is believed:

(1.) That the ore-deposits were formed through a molecular substitution and replacement of the dolomitic limestone by hot alkaline mineral-bearing solutions coming from *below*, or by metal-bearing waters and vapors from solfataric action following the ejection of the dyke porphyries, and consequent thereupon.

(2.) That such ascending solutions more readily penetrated the limestone along the planes of contact of the igneous and sedimentary formations, and through zones of least resistance in the latter previously marked out by the intrusion of the porphyries, and gradually replaced the limestone with their metalliferous contents in the form of sulphides.

The first and second of the above statements need no further comment, as the ore-bearing features of the blue limestone are now as

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well understood and generally accepted as was the upper-contact belief of the early day, and of the fourth, also, further discussion appears unnecessary, as it is fully substantiated by the developments shown upon the sections and map.

On the theory of the origin of the ores, however, volumes have been and will be written by the adherents of different schools, and these will often differ very widely, but it is generally agreed, and cannot be refuted, that the ore-deposits, whether of Iron Hill or the entire Leadville district, were deposited through a metasomatic substitution and replacement of the limestone by mineral-bearing waters. Further than this, and as to the source from which such solutions derived their metallic contents, and how they were brought within the limestone, are questions involving differences of opinion which are much more speculative and difficult to decide.

In the able discussion of the genesis of the Leadville ore-deposits by Mr. Emmons, it is held that the metallic contents "came from above"* and from a segregation of the porphyry bodies which occur at horizons above the blue limestone.

It is evident that the conditions upon which this explanation of Mr. Emmons for the origin of the Leadville ore-deposits was based, was that the upper horizon of the blue limestone in contact with the white porphyry was the principal plane of ore-deposition as is clearly stated and explained by him. Furthermore, it is seen that Mr. Emmons in presenting this theory for the origin of the Leadville ore-deposits, entertained some misgivings as to its soundness under the light of future and deeper developments, for he says: "It is possible that in future years, when mine-workings shall have been extended over areas where the ore-horizon exists at considerable depths below the surface, and other eruptive channels have been found and critically examined, evidence may be obtained that ore-solutions have ascended along these channels from below."† With the advantages now given for study and investigation of this subject by the extensive development-work in Leadville mines, this possibility appears to be verified, and the theory of the origin of the ores, from segregating and descending solutions from the white or other porpyhries is now scarcely tenable.

In the first place it is clearly demonstrated that the overlying white porphyry exerted no derivative influence whatever upon

* Emmons, *Geology and Mining Industry of Leadville*, p. 379.
† Emmons, *Geology and Mining Industry of Leadville*, p. 584.

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the ore-formations except in their secondary or oxidized conditions. If such had been the case, the upper contact of the limestone would have been undoubtedly, as then supposed, the horizon of principal substitution and deposition of the ore-bodies. On the contrary, this is now found to be erroneous. In fact, the very reverse is the case. The largest ore-bodies yet disclosed in the A. Y., Minnie, Mikado, and Silver Cord mines, and upon the second and third contacts of Carbonate Hill, have only the most remote, if, indeed, any such association whatever, with the overlying white porphyry. It would also seem probable that, if such solutions derived by segregation their metallic contents from the porphyries in percolating through them, cavities of more or less magnitude, filled with ore, would be found within the porphyry mass itself, or within its cracks, joints, and fault-fissures. With the exception of a few gash-veins or deposits of recent formation near the surface, consisting of oxides of manganese and iron, and containing but a few ounces of silver per ton, this has never been disclosed in the mass of the white porphyry. In the deeper levels of the mines the porphyries are found contiguous to the ore-bodies entirely unaltered, and showing no decomposition or trace of oxidization of the iron pyrites, as is very common nearer the surface.

Again, as the result of decomposition and substitution of the Carboniferous limestone, large masses of dolomitic sand, called by the miners lime-sand, are frequently found filling the planes of the upper contact, and overlying the ore-bodies in the chutes, as, for example, in the Iron mine, in the Gold-ore chute of the Silver Cord, and within the Fryer Hill ore-deposits. If such ore-bearing solutions came originally from above, by percolating downward through the porphyry, the substitution of the limestone and consequent ore-deposition should have been most complete and uniform upon its upper horizon, and nearest its white porphyry contact, and such replacement should appear greatest from the top of the limestone downwards. It would also appear that, if the descending ore-bearing solutions derived their metalliferous contents from the leaching of these porphyries, such porphyries must have all contributed, in proportion to their initial mineral contents, to charging the ore-conveying solutions, and that the resultant ore-deposits should have differed considerably from each other in proportion to the different constituent parts of the mass from which they were derived. Such variations have not been observed anywhere within the limestone belt. All of the deposits show themselves to have been contempo-

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raneously formed, originating from the same source and formed in the same general manner. If derived directly from these porphyries, it is most probable also that some particular intrusion must have been of itself principally instrumental in their deposition. The three principal porphyries most closely associated with the Leadville ores Mr. Emmons defines as the white or overlying porphyry, the pyritiferous porphyry, and the dyke or gray porphyry—here stated in the order of their greatest magnitude of flow and consequent importance to the leaching theory. In the deeper developments of the Leadville mines these three porphyries may all be examined in their undecomposed, unoxidized, and apparently unchanged state. As to the first or white porphyry, the flow of greatest extent and volume, occurring uniformly above the ore-horizons, Mr. Emmons's conclusion, that this porphyry was their source or origin, seems to be based upon rather insufficient data. Out of eleven specimens of white porphyry analyzed by him, he finds only three containing most minute quantities of silver, these three coming from dykes. None contained gold or zinc, and all contained oxides of manganese. With such small traces as these, which can be found in both the igneous and the sedimentary rocks of this district, it appears rather difficult to base conclusions for the origin of the enormous ore-deposits found in the Leadville region; especially as it must be considered that so complete and uniform a process of leaching as is stated by Mr. Emmons and considered in this hypothesis is scarcely probable; and that such a mechanical and chemical phenomenon of separation should leave the rock-mass in its apparently original condition without local and isolated deposits, and only a trace of the precious metals remaining, is difficult to imagine. From a comparison of Mr. Emmons's analyses of the different porphyries, it seems that the younger and lesser intrusions of gray and pyritiferous porphyries contain the greatest values in metallic contents. If, then, the segregating or leaching process has robbed these of their contents, to be conveyed through the medium of percolating waters and deposited in the limestone by a metasomatic interchange of particle for particle, then such removal has been remarkably and uniformly accomplished, through the entire mass of this rock, irrespective of its constantly changing position and thickness, and through all of the complex ramifications incident to its intrusion. It may be argued that all of the metallic contents of the white porphyry have been leached out and re-deposited, while with the younger porphyries, which show the most appreciable silver

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values remaining, the same process has been incomplete, and but partially accomplished. In this regard it would seem plausible that the thinner and smaller flows of igneous rock should be more readily penetrated by infiltrating solutions, and more thoroughly leached than the thicker and larger masses, and yet the latter still show the most considerable quantities of gold, silver, lead, iron, and zinc remaining. Another observation which will have some bearing upon the relative merits of the theories of ascending or descending solutions, as related to the origin of these ore-deposits, is the investigation of the manganese and zinc found in the ore-deposits themselves.

In the previous portions of this paper the significant occurrence of the hydrated oxides of manganese has been cited, showing the deposits of the manganese filling or wad to be most extensive nearest the surface, and near the upper limestone contact, also filling the cracks, crevices, etc., of the white porphyry, and by its dentritic stains upon its cleavage planes giving rise to the local name of forest rock. In the analyses so far made of the white porphyry, all of them show appreciable quantities of MnO and MnO_2. Within the altered or carbonated ores on Iron Hill, these oxides of manganese are of common occurrence. It is seen they become less prevalent with increase of depth and finally, upon reaching the limits of the action of surface-waters or the primitive unaltered sulphide ores, the manganese practically disappears. It may be safely stated, from the results of a careful and continued search for it, and from the corroborative evidence of chemists, assayers, and smelters of Leadville, that only comparatively small traces of manganese, in any form, have ever been found in the sulphide ores of this district, these traces being the result of a secondary infiltration and deposition from above. In other localities, and notably in the Robinson mine, and others of the Ten Mile district, its association with the sulphides in the form of a carbonate of manganese is of common occurrence. From the above observation the inference would be, that in the original deposits of the ore-bodies the manganese formed no part of them from whatever source they may have emanated, but was the resultant deposit of distinctly different mechanical and chemical action, from that which caused the original sulphide ores. For had the ore-bodies of Leadville owed their origin to the segregation and leaching of the white porphyry, the manganese existing in the porphyry in large quantities should have been deposited in the same manner as the ores, and, at the same time, as a constituent part of the original sulphide ores,

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which does not appear to be the case. It is believed that from the mode of occurrence and condition of manganese as found in the carbonated ores, that it was of secondary and even recent origin, having been distinctly the product of leaching action principally by surface-waters permeating the white porphyry, which kept pace with the porphyry decomposition and erosion, filling the cracks and tilted contacts of the limestone, and depositing itself preferably in the ore-bodies as a filling for the decreased bulk occasioned by the shrinkage and loss due to the oxidized condition and alteration of the sulphide ores.

Again, as regards the zinc, none of Mr. Emmons's analyses of the Leadville porphyries shows any traces of this metal. Its absence, however, in instances where he finds but traces of lead and silver upon which to base his skilful deductions, might be accounted for by the more ready solubility of zinc sulphide than the sulphides of the other metals, and its consequent apparent obliteration from the porphyry body would be due to its greater solubility and removal. At the same time, this, in connection with the observed removal and re-deposition of the zinc from the original sulphide ores in the process of alteration to carbonates, would give rise to the belief, that, in traversing and permeating immense bodies of the porphyry, the solutions would have re-deposited in cavities most favorable for arresting and retarding the circulation of such metal-bearing solutions some of their zinc contents, and left more definable traces in the porphyry than can be discovered, even in proximity to the largest ore-bodies.

Finally, on page 573 of Mr. Emmons's report will be found the following, very pertinently stated: "As at the time of deposit the sedimentary rocks . . . . . were horizontal and relatively undisturbed, if the deposit had been made from ascending currents, it would naturally be expected that the process of deposition should have acted from the lower surface of the beds upwards instead of from the upper surface downwards, as shown to have been the case in the blue limestone which carries the bulk of the ore."

And again, "the principal water channel at the time of deposition was evidently the upper contact of the blue limestone with an overlying porphyry, and from this surface they penetrated downwards into the mass of the limestone. It may be assumed, therefore, that the currents were descending under the influence of gravity rather than ascending under the influence of heat."

As before stated, the above quotations show that the foundation

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upon which the theory of descending currents is based was solely the accepted and entirely erroneous idea of contact-deposits upon the upper horizon of the limestone, as the distinctive feature of the Leadville mines. With the present better opportunities for investigation, these premises upon which the arguments for the descending currents were based, are shown to be in error, and if they do not entirely upset, at least they weaken the strength of the theory as applied to this locality. The above reasoning of Mr. Emmons would now seem to strengthen the opposite theory, the theory of ascending currents, from the fact that the ore-deposits *are larger* from the lower horizons and contact-planes of the blue limestone upwards, and the upper contact of the limestone with the white porphyry is not the principal channel and plane of deposition of the ore-bodies. It therefore does not seem possible, in considering the results and bearing of present developments upon the origin or genesis of the Leadville ores, that the mineral solutions from which they were formed came from above, or derived their mineral contents from segregation of the eruptive rocks.

Did the solutions come from below? I have only endeavored, in the preceding pages, to present briefly a few of the reasons founded upon existing facts and observations which would, in my opinion, make the theory of descending solutions and the leaching of the porphyries untenable at the present time as an explanation for the origin of the Leadville ore-bodies. I am aware that it is easier, in practical or in scientific geology, to pull down than to build up, and that the showing of real and supposed fallacies in one theory does not necessarily establish another. Moreover, a number of the facts here presented would harmonize with either the theory of descending or ascending currents; for example, the marked and self-evident relation between the gray porphyry intrusions and the ore-deposits may have been the result of purely physical conditions, rather than of mechanical and chemical action combined. These intrusions, by their disturbance and splitting of the limestone strata, tracing out the lines of the easiest substitution and deposition, would be equally favorable for saturated solutions carrying metallic contents derived from the leaching of the porphyries above, as for mineral-bearing solutions accompanying solfataric action, and bringing their mineral contents from below. In the case of the contact ore-bodies, however, underlying thin sheets of gray porphyry, and not connected with the white porphyry above, the ore-deposits should have been formed on top rather than below such thin sheets, if such solutions

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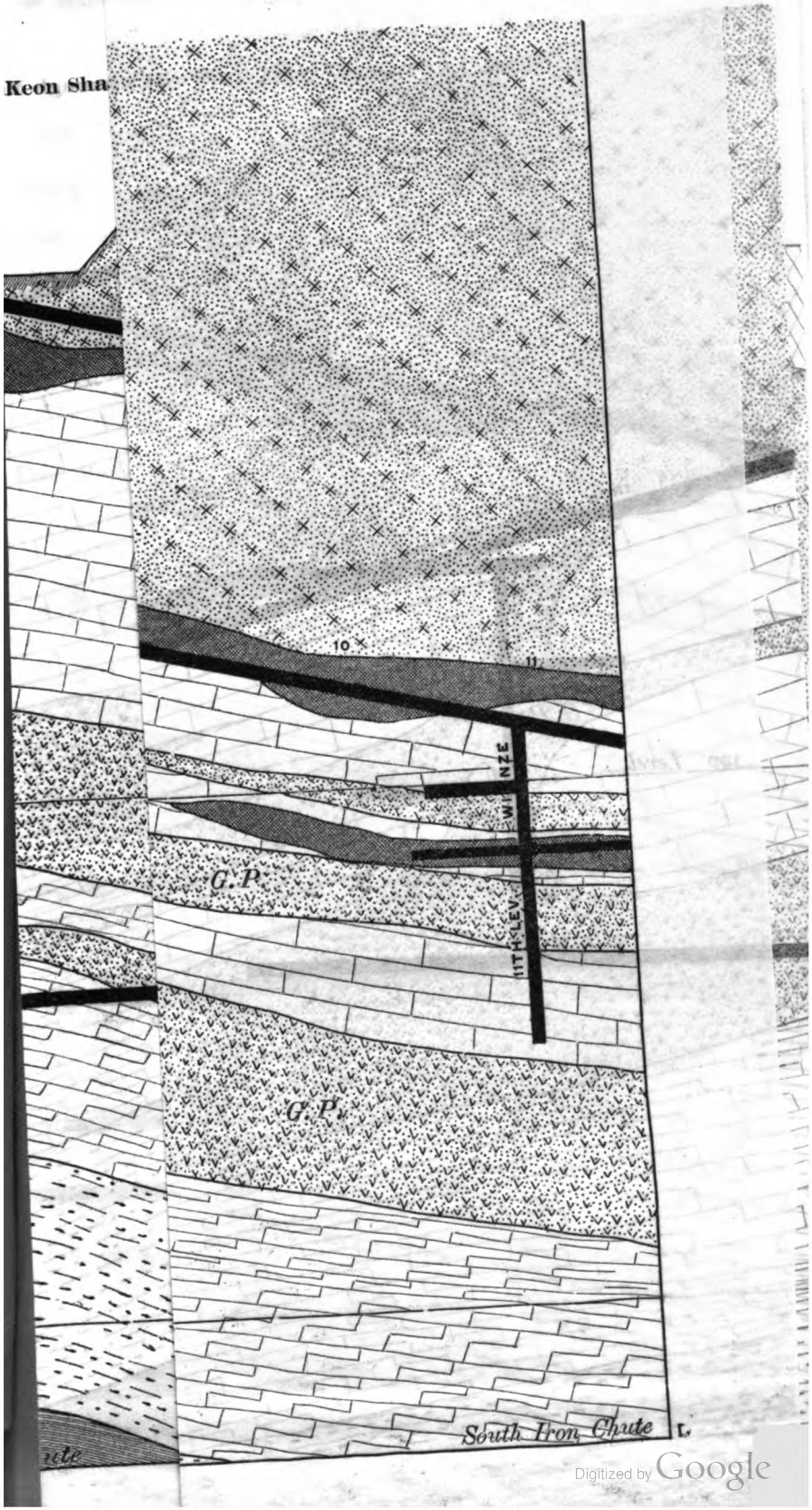
Plate III.—BLOW.



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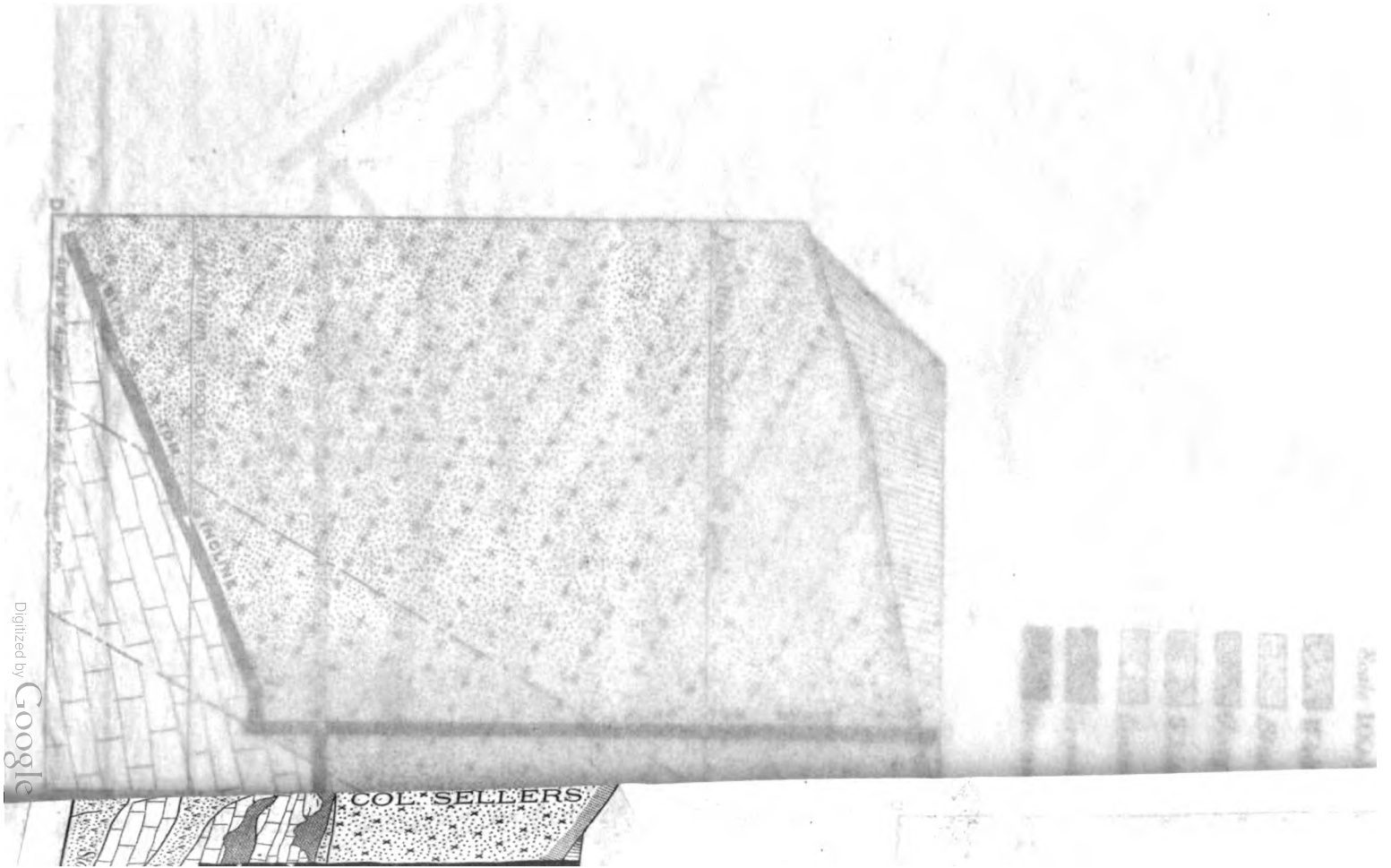

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Plate IV.—BLOW.

LEGEND

- *White Porphyry*
- *Blue Limestone (Carboniferous)*
- *Gray Porphyry*
- *White Limestone (Silurian)*
- *Lower Quartzite (Cambrian)*
- *Granite (Archaean)*
- *Indicates Side lines of Claims*
- ,, *Shafts & Levels*
- ,, *Faults*
- ,, *Ore & Vein Material*

Scale 200 ft. to 1 Inch.

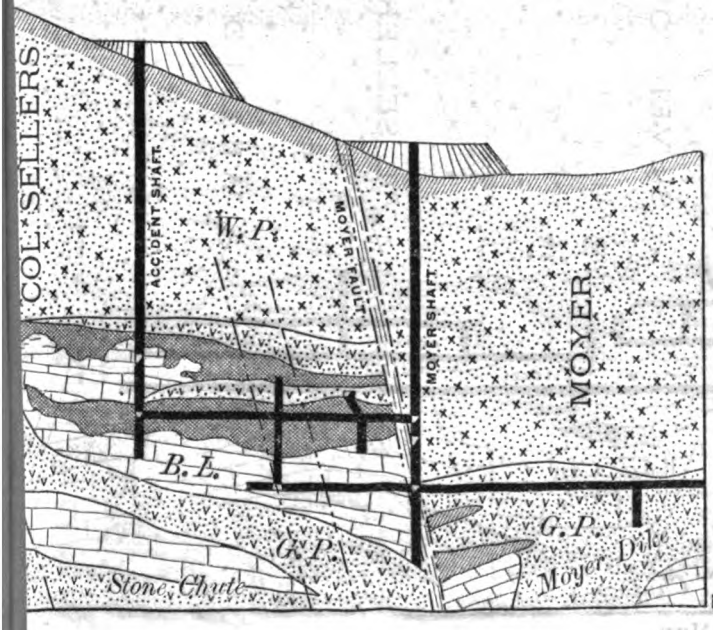

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Plate V.—BLOW.

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came from above downwards. This is known to be not generally the case. Moreover, the belief in solfataric action as the source of the ore does not necessarily preclude the theory that the porphyries were the derivative rock-mass from which the ascending solutions receive their metallic contents, since they may have been segregated from the decomposition of the porphyritic magma by subterranean agencies of heat and pressure, and thus charged with their contents of the precious metals to be ejected subsequent to the porphyries, and consequent upon their intrusion. To demonstrate that the solutions came from below is, as Mr. Emmons states, not susceptible of direct proof, from the impossibility of obtaining data bearing upon the purely conjectural conditions and forces to which such an hypothesis would be subject. If, however, from the negative course of reasoning taken, it can be shown that the origin of the metallic contents of these solutions was not from the porphyry above, or from a leaching from any of the neighboring rocks, without going into other theories of vein-formation, which, it is conceded by all authorities, would not apply to the limestone-ores of the Leadville district, it would appear, then, that there was but one other source from which the metallic contents of such ore-depositing solutions could be derived; that is, from below, and that such ore-deposits were formed from hot metal-bearing solutions and vapors, which solutions were forced up with their mineral contents from the deep, from the same regions, and in the same manner as that of the intrusive dyke porphyries, with which they are here and elsewhere so closely and intimately associated.

NOTE.—In Plate I, "Map showing the Ore-chutes," etc., the stope shown around the Gross shaft is an error. There is no such stope at that point; the line shown as the boundary of the stope should be simply the 11,040 feet contour, and the ground within it should have been left unshaded.

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IMAGINARY BOUNDARIES.

BY R. W. RAYMOND, NEW YORK CITY.

(Colorado Meeting, June, 1889.)

IN my paper on "End-Lines and Side-Lines in the Mining Law," read at the New York meeting of February, 1889 (*Trans.*, xvii., 787), I discussed certain points involving the rights of a locator, B, who had located and surveyed a piece of ground, X + Y, of which only Y was properly subject to location, X being already held by another claimant, and had applied for and received a patent, granting X + Y, "excepting and excluding X." The paper concluded with the following questions:

> "Is there any difference in effect between the grant to him of Y simply and the grant of X + Y, 'excepting and excluding X'? Why should any mining locator stake out his boundaries upon another location, and thus appear to assert a claim which he immediately withdraws? What is the legal force of such boundaries, established upon the land of another?"

After calling attention to the fact that the entrance upon another's land for such a purpose is a trespass, and the principle that no trespasser can create by his trespass any right which he would not have without it, I reserved to another occasion a review of the history of the absurd practice of imaginary boundaries for mining claims, and a discussion of their legal effect.

This practice is founded, of course, in the regulations and decisions of the United States General Land Office. There is nothing in the Revised Statutes which hints at it. On the contrary, the terms of Section 2322, giving to a locator "the exclusive right of possession and enjoyment" of the surface of his claim, seems clearly to forbid the use of that surface for the boundary-stakes of another locator. But the Act of 1866, though it provided for the issue of patents upon payment to the Government of so much per acre, and really by implication (as I have always believed) involved the absolute grant of the surface, was not so construed by the courts generally, or by the Land Office. Surveys were laid one over another in inextricable confusion, and the United States sold at $5 per acre the

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same land over and over again, without passing any valid title to it. In other words, an easement was sold by the acre, and overlaid with another easement upon the same acre.

When the Act of 1872 substituted for this vague basis of title the fee simple of the surveyed claim, and the extra-lateral right attached to apexes within it, the General Land Office still retained, in its regulations and in the form of its lode-patents, some of the phraseology of the earlier law. Thus, as I have shown in the paper cited, a patent issued in 1884 granted not only a given tract, but *also* "—— linear feet of the said —— vein, lode, ledge or deposit, for the length hereinbefore described," etc. This additional grant was without meaning or force. The new law authorized the conveyance of neither more nor less of the linear extent of the located lode than was embraced within the location. This location must have parallel end-lines, and must not exceed 1500 feet in length along the vein. But a location in rectangular form, 1500 feet in length, might contain more than 1500 linear feet of a vein curving through it, without crossing its side-lines, and no one can pretend that the whole length of the lode within such a claim would not be covered by it.

The question of the shape of a location-survey was considered in a letter of Commissioner Williamson to F. C. Morse, Fairplay, Colo., dated January 21, 1879, from which I quote the following:

" . . . . You also submit a diagram in the form of a triangle, and ask whether such surveys will be approved by this office. On this point I would say, that in no case can a triangle, *which embraces the entire lode or vein claimed* (*a*), be approved, unless the lode itself extends into and fills the point in the acute angle, and then only when adverse rights (*b*), existing on the 10th day of May, 1872, render it necessary. See Section 2320, United States Revised Statutes (*c*), which precludes a restriction of the width to less than 25 feet on each side of the middle of the vein at the surface.

"Neither can the surface-ground (*d*) extend beyond the end of the lode in any instance.

"Where a lode intersects another claim, and extends within a prior survey or location, it may be patented to the length allowed by law (*e*), and if the end of the lode is found within such prior location, the surface-ground may close upon the prior survey, *provided* (*f*) the extension of the end-line within such prior survey, *parallel to the other end-line*, would not exclude any portion of such surface-ground.

"Where a survey of the kind last above mentioned results in a triangle formed against the prior survey, the shape of the same, subject to the restrictions as to the surface-ground extending beyond the lode before named, will not be objectionable (*g*), but this will not apply where the lode embraced by the subsequent survey is merely the extension or continuation of the same lode on which prior location is made, for in such case (*h*) the lode itself cannot extend within the prior location."

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For convenience of reference, I have introduced into the foregoing quotation letters to correspond with my observations upon its various parts. It is extremely difficult to discover either its meaning or its reasons, as will appear upon any attempt to analyze it.

(*a*) The italics are those of the Commissioner, and are apparently intended to emphasize the condition stated. The idea seems to be, that if the whole length of a lode be included in a claim, *i.e.*, if the lode does not strike into an adjoining claim, the location may not be a triangle unless the lode extends into and fills the acute angle, and then only, etc. But the Commissioner could not have been ignorant of the frequent impossibility of knowing, at the time of making a location, whether the lode extends beyond it or not. If the adjoining ground is that of a prior claim, it is not lawful for the new locator to explore within it. We are forced to conclude, therefore, that in this italicized clause the phrase *lode or vein* should be *length of lode;* so that the meaning is, that a *whole claim* cannot be a triangle, except as stated. This is confirmed by the final passage (*g*), which declares that part of a claim may be a triangle—of which, more hereafter.

(*b*) This limitation as to date does not at all apply to the case, as will be seen below. What does apply, is the positive declaration of Section 2320 that the end-lines of each claim shall be parallel to each other. Since this cannot be true of a triangle, it follows that no claim located after May 10, 1872, can legally be a triangle, adverse rights of whatever date to the contrary notwithstanding.

(*c*) Section 2320 is here grossly misquoted. It prohibits, not claims narrower than 25 feet on each side of the vein, but the limitation of claims to that effect by any mining regulation (except when rendered necessary by vested rights of prior date). In other words, no local regulation shall oblige a locator to make his claim less than so wide, if there was, on May 10, 1872, room for that width of claim at least. But the locator himself might and may, for all that Section 2320 has to say on the subject, stake out as narrow a claim as he chooses. Or he may make it wide at one end and narrow at the other, provided its two end-lines are parallel. To take the case of a triangle, with the lode entering the acute angle and crossing the base, it would be only necessary to cut off the tip of the triangle, giving ever so short an end-line parallel to the base, in order to have a perfectly legal claim. I repeat that the passage about existing adverse rights applies explicitly to compulsory restriction by regulation, and not at all to voluntary restriction by the locator himself. It leaves

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him perfectly free to do what it protects him against being forced to do.

(*d*) This is another obscure proposition. In this and succeeding passages the Commissioner talks quite fluently about the "end of the lode," as if it were a very simple matter for the United States mineral land surveyor to find it and fix its position. Logically, one might be tempted to infer that affirmative proof of the crossing of both end-lines by the apex of the located lode would be required before approval of the location. Perhaps it might be deemed to follow also that the surface-rights, as well as the apex-rights, on a located lode would end at any place where the apex should escape from the location across a side-line. But such has not been the practice. A claim once being duly located and properly held or patented, remains good as to the whole surface, though the lode leave it midway. This passage in the Commissioner's letter seems, therefore, to mean merely that a claim cannot be located so as to encroach on a prior location unless it is laid along a lode which runs into that prior location. On which the obvious comment is: "Nor then, neither!"

(*e*) This is scarcely mistakable. The "length allowed by law" is any length not exceeding 1500 feet; and the declaration is here plainly made that a lode may be claimed and located into a prior claim, only the overlapping surface will belong to the prior locator. But so will the apex and all the mineral beneath it, and the extra-lateral right thereon depending.

(*f*) This proviso could not apply to a simple rectangular claim, I think. It would, however, prevent the acceptance of such irregular forms as that of the Stone claim in the Elgin case (*Trans.*, xv., 292).

(*g*) The exterior triangle, which may thus be left as the sole surface-ground of the later locator, is now declared to be, under the restrictions named, not objectionable. And the next following passage indicates, in the mind of the Commissioner, an impression of some sort that the locator, "to the length allowed by law," acquires, even within the prior claim of another, some right to the part of that length not covered by his own surface-title. There need be but one criticism upon this proposition. It is not true.

(*h*) This declaration sounds absurd. What it probably means is, that a lode other than the one which was made the basis of a prior location may be "claimed" within that location by a later comer; but of course the located lode itself cannot be so claimed, the whole length of it within the said prior location having been claimed

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already. The exception here made should be stated as the rule, and the rule should not be stated at all, being a pure assumption directly contrary to the statute; for the law makes no difference as to ownership by a locator between the discovered and located lode and all other lodes apexing within the claim.

The assumption pervading this letter seems to have controlled also the whole practice of the General Land Office down to December, 1884, when the following circular, issued by Commissioner McFarland and approved by Secretary Teller (himself a lawyer, familiar with controversies arising under the mining law), essentially altered the conditions imposed upon locators. I give this important document in full, marking, as before, with italic letters, the passages to which I shall call special attention below:

DEPARTMENT OF THE INTERIOR—GENERAL LAND OFFICE.

To U. S. Surveyors General and
Registers and Receivers.

Washington, D. C., December 4, 1884.

Gentlemen: 1. The rights granted to locators under Section 2322, Revised Statutes, are restricted to such locations on veins, lodes or ledges as may be "situated (*i*) on the *public domain*." In applications for lode-claims where the survey conflicts with a prior valid lode-claim or entry, and the ground in conflict is excluded, the applicant not only has no right to the excluded ground, but he has no right to that portion of any vein or lode the top or apex of which lies within such excluded ground, unless his location was prior to May 10, 1872 (*j*). His right to the lode claimed terminates where the lode, in its onward course or strike, intersects the exterior boundary of such excluded ground, and passes within it.

2. The end-line of his survey should not, therefore, be established beyond such intersection, unless it should be necessary so to do for the purpose of including ground held and claimed under a location which was made upon public land and valid at the time it was made (*k*). To include such ground (which may possibly embrace other lodes) (*l*) the end-line of the survey may be established within the conflicting survey, but the line must be so run as not to extend any further into the conflicting survey than may be necessary to make such end-line parallel to the other end-line, and at the same time embrace the ground so held and claimed. The useless practice in such cases of extending *both* the side-lines of a survey into the conflicting survey, and establishing an end-line wholly within it, beyond a point necessary under the rule just stated, will be discontinued.

3. These instructions will be observed by surveyors-general in all cases where surveys have not been approved by them prior to receipt hereof.

4. If, however, a survey under the old practice has been approved by the surveyor-general prior to the receipt by him of these instructions, application for patent thereon, if otherwise regular, will not be rejected.

5. In applications filed prior to receipt hereof at the local land office, and applications allowed under the preceding paragraph, entry will be allowed as heretofore, when the necessary proofs under former regulations are complete.

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6. In case of applications and entries allowed under paragraphs 4 and 5, amendment of the survey will be directed by this office, if found necessary.

7. After the receipt of this circular at the local land office, all applications for mineral patents, applications to purchase, register's final certificates of entry, and receiver's receipts must not only describe the ground claimed, but must state specifically what conflict or conflicts with other surveys, lots or claims are excluded, giving the number of each conflicting survey or lot. The published and posted notices must contain the same information.

8. As this circular does not affect any rights which an applicant has under the law (*m*), its enforcement in pending cases cannot operate injuriously, and it will therefore be carried into effect at once in the adjudication of cases by this office. In the form of patents to be issued, the same rule will go into operation as soon as the necessary blanks and records can be prepared.

9. A strict observance of these regulations will be required.

Very respectfully,
N. C. McFARLAND,
Commissioner.

Approved.
H. M. Teller,
Secretary.

Before offering any comments upon this circular, I will give the substance of an important ruling made in the same connection March 19, 1889, by Mr. Noble, Secretary of the Interior, upon appeal from a decision of Commissioner Stockslager, of the General Land Office, and also the substance of the decision itself, from which the appeal was taken to the Secretary.

The case is that of Mineral Entry No. 1038, made at Lake City, Colorado, on the Eldorado lode-claim, the survey of which overlapped a prior location, the overlapping portion being excluded from the Eldorado entry and application. The Eldorado survey was made long after the circular of December, 1884, reforming the old practice, had been received by the local officers; but it intruded upon the adjoining senior location in the old-fashioned way, including within imaginary boundaries upon the said adjoining ground an area to which it could have no right, and which was explicitly excepted.

The Land Office directed a new survey of this part of the location, omitting the overlapping area, except so far as permitted by the circular of December, 1884. From this decision the applicant appealed, contending that Section 2336 of the Revised Statutes gave to the junior location certain rights which the Land Office could not limit. This section provides that "where two or more veins intersect or cross each other, priority of title shall govern, and the prior location shall be entitled to all ore or mineral contained within the space of intersection, but the subsequent location shall have the right of way

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through the space of intersection for the purposes of the convenient working of the mine." In the paper on end-lines and side-lines, already cited, I have given at length my view of the meaning of this section, and my reasons for holding that it cannot apply to intersections of locations, and hence that it is inapplicable to intersections of veins *in strike*. Secretary Noble, in his decision upon the foregoing case, takes a different view, although, in this instance, with similar ultimate result.

I quote from Copp's *Land Owner*, May 15, 1889, p. 40, the pertinent passage:

"In the case at bar the survey overlaps but does not cross or intersect the prior location. It cannot, therefore, be held to come within the purview of Section 2336, the manifest purpose of which is to provide for the working of such claims as cross and lie on both sides of a senior location or entry (*n*).

"In the case of Branagan *et al. vs.* Delaney (13 C. L. O., 190), upon which the appellant relies, the Supreme Court of Colorado considered the respective rights of a junior and senior location which crossed each other.

"In the present case, no part of the overlapping space is embraced in the Eldorado application or entry, but said space is expressly excluded therefrom. . . . . .

"The appellant's right does not extend beyond a southerly end-line (parallel with the north line) through the point where the lode intersects the exterior line of the said senior location (*o*). The surface-right being simply an adjunct to the lode-claim (*p*), such right could not extend beyond the same. It was therefore proper to require the end-lines of the survey to be readjusted so as to accord with the requirements of the law and the regulations." (Decision affirmed.)

(*i*) This statement is substantially accurate, though, strictly speaking, it is not such *locations*, but all locations made "on any mineral vein, lode or ledge, situated on the public domain," which constitute the subject of the section. But the purpose of the Act of 1872, as embodied in this section, was to relieve mining titles from just such difficulties as arise from their dependence upon obscure, technical or still undiscovered conditions. Where a lode is "situated," it might be hard to say; but the situation of a location, which is a tract of land, can be easily and permanently fixed. Hence, the law makes all the mining rights depend absolutely in one way or another on the lines of the surveyed location; and the Commissioner's construction is correct.

(*j*) This exception is necessitated by the construction of the law of 1866, according to which a location made, and even patented, did not remove from the public domain the tract it embraced.

(*k*) This provision covers locations made according to the old

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practice, but not surveyed before the promulgation of this circular. In such a case, a location laid out, for instance, in the form of a rectangle, entering a prior location obliquely, and having one of its parallel end-lines therein, would, if reduced to its proper limits, exterior to the prior location, fail to comply with the requirement of parallelism in end-lines. The only way in which such parallelism could be secured without surrendering a part of the validly-claimed surface would be to permit the new end-line, and part of one side-line, to be located within the prior claim. It will be noticed, however, that the circular does not permit this to be done in locations to be made after its date.

(*l*) The very proper reason here given for permitting it in the case of earlier locations is, that the locator may be protected in the ownership of the whole of the surface so located on the public domain, and not forced by a new ruling of the Land Office to surrender what may be a valuable part of it.

(*m*) The wisdom of this concession is shown in this paragraph, which effects the immediate enforcement of the new practice, without danger of embarrassments arising from vested rights under the old.

(*n*) This is a curious misreading of the statute, which says nothing about crossing of *claims*, but deals wholly with *veins*. Under the present law, that is, since 1872, claims cannot possibly be located so as to cross one another. Such claims located earlier than 1872 would indeed present a condition to which this section is applicable; for in that case both claims might be valid as to their common space. And the exclusive grant made by Section 2322 to the prior locator would not carry the right to all that portion of the lode of the junior locator lying within the prior claim. For the said grant is coupled with the condition "where no adverse right exists on the 10th day of May, 1872," and the lode-right of the junior locator in the case supposed is such an adverse right. But with regard to all locations made under the law as established in 1872, the junior locator can have upon that portion of his lode, the apex of which lies within the senior location, no rights at all—not even the right of way. As to claims located before 1872, and containing lodes which intersect on the strike (that is, the apexes of which intersect), Section 2336 does apply to the space of intersection of the veins. As to all claims of whatever date it applies to the intersection of veins on the dip. If under the present law a location should be attempted completely crossing an earlier one, the Land Office ought not to accept it as a

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valid one. The locator should be required to make two discoveries and two separate locations.

(*o*) On careful examination of this requirement it will be seen to depart from the rule laid down in the circular of December, 1884 (see *k* and *l* in the circular above). The difference will appear upon a study of the diagram, Fig. 1.

It should be premised that the Eldorado location, involved in the Secretary's decision, having been located in 1877, comes under the proviso (*k*) in the Commissioner's circular.

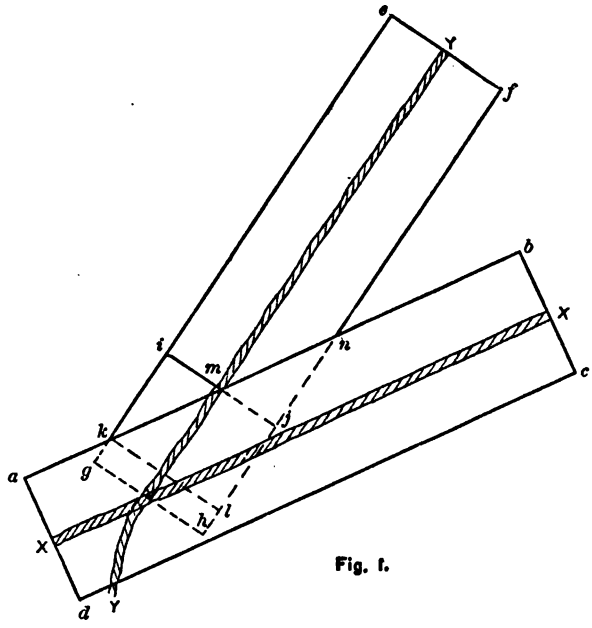

Fig. 1.

Let *a b c d* be the senior location upon the lode, *X X*, and *e f g h* the junior location on the lode, *Y Y*. This lode crosses the side-line, *a b*, at *m*, while the two side-lines of the junior location cross the senior side-line, *a b*, at *k* and *n* respectively. According to the terms of the Commissioner's circular, the surveyed end-line, *g h*, is erroneous, and a new end-line must be drawn parallel to *e f* and not extending any further into the senior location than may be necessary to make such parallel end-line embrace the ground validly held and claimed under the location, *e f g h*, when the same was made upon public land. It seems clear that this ground is the polygon, *e f n k*, and that the permissible end-line is *k l*, the only one

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which can be drawn parallel to *e f* at a minimum distance from *m*, and including within the corrected survey the whole of *e f n k*. The sentence (*k*) in the circular: "The end-line of his survey should not, therefore, be established beyond such intersection" [in this case, *m*] "unless it should be necessary so to do," etc., intimates that it might be necessary, and applies to the case in the diagram.

But the Secretary's decision (*o*) requires the new end-line to be drawn through the point of intersection, *m*. In one case, the approved location would be *e f k l*, and the grant of the surface would except *k n l*, already granted to another. In the other case, the approved location would be *e f j i*, and the grant of the surface would except *m n j*. The rights on the lode, *Y Y*, of the junior locator would be the same in either case, because one location gives him as much of the apex of that lode as does the other. But he would, in the latter case, lose the surface-right to *i m k*, contrary to the praiseworthy consideration shown in the Commissioner's circular (*l*), which observed that such ground "may possibly embrace other lodes."

(*p*) Here we have the Secretary's reason for the requirement of the new end-line, *i j*. It is, that the triangle, *i m k*, was never validly claimed, and that the locator should therefore not be protected in his possession of it, because "the surface-right, being simply an adjunct to the lode-claim, such right could not extend beyond the same."

Is the surface-right "simply an adjunct to the lode-claim?" It seems more accurate to say that the lode-right is an adjunct to the surface-claim, since that is the way the statute puts it, granting "the exclusive right of possession and enjoyment of all the surface" first, and the right to the lodes afterwards, the latter being limited by the former. Moreover, the surface-right may be granted without any extra-lateral lode-rights, or the extra-lateral right may be taken away, as it was by the decision in the Elgin case, leaving the surface-right intact. The latter can scarcely be called an adjunct if it can thus subsist alone.

On looking more closely, however, we find that the persons to whom the grant expressed in Section 2322 is confined are locators of mining locations "on any mineral vein, lode or ledge." Before a surface-right can be acquired, there must have been a "lode" discovered, and the location must have been made "on" that lode. Evidently the Secretary holds that wherever the right to locate on that lode ceases, there the right to make a surface-location ceases. Thus, in Fig. 1, the junior locator could not claim the lode, *Y Y*,

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beyond *m*. Consequently, it is argued, he could not properly claim any surface beyond a line through *m* parallel to his other end-line, *e f*. If he desired to acquire the surface-area, *i m k*, he should have made the other end-line parallel to *k n*. There seems to be some hardship in this. For instance, suppose the line, *e f*, were already the boundary of a prior location, so that *e f n k* were the only piece of the public domain left unlocated in that locality. Evidently, under this ruling, the junior locator could not possibly get the whole of it. However he might lay his lines, they would leave a triangle of unoccupied ground at one end or the other. Such remnants, as miners well know, are the bane of mining titles. Whether they are located (as they often are) by piratical intruders or not, they are, at least, open to the free exploration which is permitted on the public domain. Apparently there would be no way for the locator of *e f j i*, in Fig. 1, to protect himself against such a nuisance except by trying somehow to find a lode in *i m k*, and get a location on it. On the whole, the method laid down in the Commissioner's circular, which absorbs this fraction into a respectable claim, would be preferable but for one consideration. The Secretary's ruling makes a location, no one boundary of which is wholly imaginary; whereas, under the Commissioner's circular, the boundary, *k l*, is entirely upon the surface of the senior location. This is almost enough to outweigh the nuisance of the triangular remnant.

It must be born in mind, however, that both these rulings have reference to the sort of survey which will be approved by the officers of the Government; not to the rights of a locator or patentee afterwards. There is no doubt, I think, that when the Government has once passed title, the surface-right of the grantee is impregnable, and no longer a mere adjunct (if it ever was so little as that) to his lode-claim.

The form of the lode-patent issued by the General Land Office has also been changed to correspond with the new practice and with the terms of the law rather than with tradition. The language of the granting clauses in a patent issued in 1884 (quoted in my former paper, cited above) was:

"The United States . . . . do give and grant . . . . the said mining premises hereinbefore described as Lot No. ——, ... with the exclusive right of possession and enjoyment of all the land included within the exterior lines of said survey not herein expressly excepted from these presents, *and of* 1500 *linear feet of the said* —— *vein, lode, ledge or deposit*, for the length hereinbefore described, throughout its entire depth, although it may enter the land adjoining, and, also, of all other veins, lodes, ledges or deposits throughout their entire depth, the tops or apexes of which lie inside. . . . ."

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The language of the corresponding part of the lode patents now issued is:

"There is therefore hereby granted by the United States unto the said . . . . the said mining premises hereinbefore described and not expressly excepted from these presents, and all that portion of the said —— vein, lode or ledge, and of all other veins, lodes and ledges throughout their entire depth, the tops or apexes of which lie inside of the surface boundary-lines of said granted premises in said Lot No. ——, extended downward vertically, although such veins, lodes or ledges in their downward course may so far depart from a perpendicular as to extend outside the vertical side-lines of said premises. *Provided*, That the right of possession to such outside parts of said veins, lodes or ledges shall be confined to such portions thereof as lie between vertical planes drawn downward through the end-lines of said Lot No. ——, so continued in their own direction that such planes will intersect such exterior parts of said veins, lodes or ledges. *And provided further*, That nothing herein contained shall authorize the grantee herein to enter upon the *surface* of a claim owned or possessed by another."

The superiority of the new form is evident. It gives what the law gives, no more and no less. But it leaves as uncertain as ever the force of imaginary boundaries. Under the old practice, as we have seen, surveys were allowed to overlap to any extent. Under the reformed practice, one corner, and either the whole or a part of one or more boundaries, may be upon the land of a prior claimant. We have therefore still to deal with the question, what these imaginary boundaries mean.

To return to Fig. 1: The senior locator on the vein, *X X*, owns also the vein, *Y Y*, from the point, *m*, where it enters his claim, and his right to follow that lode in depth is bounded by two parallel vertical end-planes, to be drawn somehow—nobody can positively say how. But one thing is certain, whatever the rights of the prior locator may be, they depend upon the relation of his own location to the lode, and are not at all modified by the subsequent drawing of the line, *g h*, or *k l*, or *i j*, by the junior locator.

Imaginary boundaries, then, can affect only the rights of the locator himself, not those of the prior locator on whose ground they are drawn. The documents above analyzed show clearly that such boundaries are considered to have no value at all in conveying title, or shadow or beginning of title, to so much of the surface they may enclose as is not at the time of location in the public domain.* It

* It may be superfluous to repeat, that under the Act of 1866 the locating of a mining claim is held not to have withdrawn the surface from the public domain. Where not otherwise specified, I wish to be understood as discussing the situation since 1872.

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follows, as I have shown in my paper on end-lines, etc., that the abandonment of the location, *a b c d*, and its consequent reversion to the public domain, would not revive any dormant rights of the junior locator on *Y Y* by reason of his imaginary boundaries within *a b c d*, for no such rights ever existed. The abandonment would simply throw open the senior location to re-location, in whole or in any part, by anybody.

Suppose the dip of *Y Y* to be towards *f h*, and the line, *i j*, to be an approved end-line, would a vertical plane on that line actually be the boundary between the undergound rights of the two locators? To make the bearing of this question clearer, I introduce another diagram, Fig. 2, in which the junior location is not rectangular. The same letters are used as in Fig. 1, so far as the two coincide.

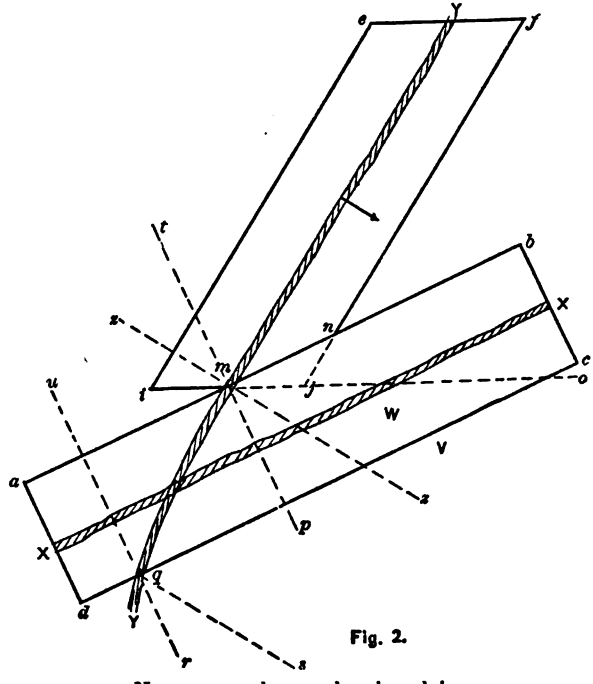

Fig. 2.

Non-rectangular overlapping claim.

In this case, *i j* does not coincide with the true dip of *Y Y*. If a boundary-plane, *z z*, is drawn on the true dip, through the point where the apex enters the senior location, it will not coincide with the surveyed line of either party. This gives us a general case, presenting four possible boundaries—*m b*, *m p*, *m z*, and *m j*. The last is

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immediately eliminated, *so far as the rights of the senior locator are concerned;* for, as we have seen, those rights originated and were defined before any junior locator existed; in other words, they cannot be in any way affected by the line, *i j*, which did not exist when they began. We have, then, three possible boundaries for the rights of the senior locator on the vein, *Y Y*, as follows:

1. The location, *a b c d*, being perfectly normal by reason of its relation to the lode, *X X*, on which it is laid throughout, the end-lines, *a d* and *b c*, are valid for *X X* and all other lodes crossing them both. But *Y Y* crosses both the side-lines; hence, by analogy of the Flagstaff decision, these side-lines become the end-lines for *Y Y*, and planes drawn through *m b* and *q c* will bound the rights on *Y Y* of the *X X* location.

I think this view is erroneous in ignoring the important fact that the Flagstaff decision deals with the located lode, and not with another lode apexing within the location. The difference is all-important; for if a location does not carry throughout its extent and across two parallel opposite boundary-lines the apex of the lode upon which it professes to be located, it does not present the conditions contemplated by the statutory definition of the extra-lateral lode-right. But if the apex does thus cross two parallel opposite sides, the condition is fulfilled, though these sides may not have been called by the locator his end-lines. Thus, if *a b c d* had been originally located upon a discovery of *Y Y*, and *X X* had been subsequently discovered, the Supreme Court might have said of it, according to the Flagstaff decision: "This location runs from *m* to *q* on the located lode, and *a b* and *d c* are its end-lines. It is too wide in the direction, *a b*. If this defect had been observed before the issue of a patent, the survey would doubtless have been rejected. As the title has passed from the United States, however, and no one but the United States is wronged by the grant, the surface-rights will stand unless the Government takes proceedings to reclaim the excess. They cannot be attacked in any collateral proceeding. But the legal end-lines of a location are determined, not by naming them, but by their relation to the located lode. Hence, planes through *a b* and *d c* bound the rights of this location." This I understand to be the Flagstaff doctrine.

It will be observed that I have introduced into this hypothetical decision a reference to the patent as barring any further change of surface-lines on the motion of adverse interests. It is true that the patent adds nothing to the rights of a locator, but it settles their

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boundaries and basis, cutting off all prosecution of adverse claims to the surface granted. Prior to the final patent proceedings, I think the Land Office might legally, of its own motion, reject as too wide such a location as *a b c d* if laid on *Y Y* as the located lode; or adverse claimants might demand its reduction to the maximum width on both sides of *Y Y* allowed by law. But I have not heard of any such case. After patent, the Government is not likely, and private parties are not competent, to interfere.

But the case is very different when the valid end-lines of a location have once been fixed by the crossing of the apex of the located lode. Vertical planes through these lines constitute the boundaries for all other lodes, as well as for the located lode, within the claim. Consequently, to return to the original proposition represented in Fig. 2, the claim, *a b c d*, located on the vein, *X X*, has, once for all, the end-lines, *a d*, *b c*; and the rights of the locator upon all lodes within it are somehow related to planes through those two end-lines. The side-lines, *a b*, *c d*, play no part in this connection.

2. Or, secondly, since planes through the end-lines, *a d*, *b c*, cannot intersect the points, *m* and *q*, which are the boundaries of the apex-right, their direction may be taken as a guide, and the boundary-planes on *Y Y* of the *X X* locator may be drawn through *m p*, *q r*.

3. Or, finally, *z z* and *q s*, which are independent of all the surveyed lines, and represent either the direction of the dip determined at *m* and assumed at *q*, or determined at *q* and assumed at *m*, or determined at both places, or averaged throughout the distance, *m q*, and thus settled, or fixed by arbitrary judgment, or compromise agreement, or jury verdict, may be the lines through which the boundary-planes are drawn. This is the principle of the Amy and Silversmith decision, discussed in my former paper, cited above, only in that case the court had an easy time determining the dip of *Y Y*, because it took the direction at *m* only. Probably, when *q* comes to be exposed, if the dip there should not be parallel to *m p*, it will be ignored. For one thing is sure—the end-boundaries must be parallel.

I do not propose to discuss this third proposition, having already expressed and explained my dissent from it. Everything that refers the miner's title to such indefinite and variable marks as the underground developments may reveal is objectionable. Proposition No. 2 seems to me preferable for certainty, convenience, substantial equity, substantial compliance with the intent of the law, and conformity with the utterances of the Supreme Court thus far.

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But what I wish to point out here is that the underground boundary of the rights of the senior locator is not necessarily the boundary also of the rights of the junior. If the plane through *a b* be the boundary, then such is indeed the case, for that plane cuts off a section which his projected end-line, *i j*, would include, and, of course, he retains all that is not cut off. But boundaries through *m p* or *m z* would not leave to the junior locator all that they did not give to the senior. For the locator of *e f i j* cannot possibly claim anything on the dip of *Y Y* not between the planes, *e f* and *i j o*. We thus have under propositions No. 2 and No. 3 respectively the triangles, *p m o* and *z m o*, which do not belong by apex-right to either locator. So far as they underlie the *X X* location, they belong to it by surface-right. Outside of that location they belong to the owner of the adjoining surface.

We reach, then, this important conclusion—that the imaginary boundaries of an overlapping location may limit the junior without affecting the senior locator. Under the new practice, as shown in Fig. 2, *i j* is the maximum limit. Under the old practice, as shown in *e f h g*, Fig. 1, *i j* is still the maximum limit, so far as the lode-rights on *Y Y* are concerned. And the junior rights may be curtailed within that limit. This would be the case if No. 1 of the propositions just discussed were accepted, and *a b* became the senior boundary-plane for the extra-lateral right on *Y Y*. I have, however, explained my opinion that No. 1 is erroneous. Under No. 2 (boundary-line, *m p*) the junior location is safe enough in the case supposed in Fig. 2, where the lode, *Y Y*, dips towards *f j;* but let the dip be towards *e i*, and the boundary, *m t*, parallel to *r u* and *d a*, cuts off a good deal that *j i* would include; and the same is true of *z z* under proposition No. 3 on a similarly modified dip.

In any case of legal conflict, in the nature of an action for trespass or ejectment or summary equity remedy, involving such boundary questions as are here discussed, the force of an imaginary boundary will be greatly affected by the relative position of the parties as plaintiff or defendant, and the position of the "burden of proof" as to title, which is determined by the pleadings. Thus, with regard to the two points, *W V*, Fig. 2, supposed to be on the dip of *Y Y* underground (it being also supposed, to avoid complication, that *X X* dips much more flatly, and the two veins do not intersect in this part of the ground, or, in other words, *X X* being ignored), it is clear that the senior locator could easily, as plaintiff, expel the junior from *W*, not by virtue of the apex-right, *m q*, but by virtue of the sur-

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face-right of the claim; for, this once proved by the introduction of the documentary evidence, the junior, as defendant, would have to produce affirmative proof of an apex-right to overcome it, which he could not do, because of the limit set by the line, *i o*. On the other hand, neither of these two parties could eject the other from *V* on the dip of *Y Y*, because neither could make positive proof of title to *V*.

No matter how far a junior claim may have been, under the old Land Office practice, extended over a senior, the legal effect of its imaginary boundaries can be ascertained by reducing it to the type, *e f j i*, Figs. 1 and 2, by drawing parallel to the exterior end-line a new end-line through the intersection of the apex of the lode with the boundary of the senior claim. In such reduced form the overlap has no further significance, and the end-line limits the locator himself, but not his senior neighbor.

How shall a case be dealt with in which both ends of a claim overlap senior claims, and both end-lines are, consequently, imaginary? I presume, by drawing two new end-lines parallel to the old through the two extremities of the apex-right of the junior claimant. It is pleasant to know that such preposterous locations will not be permitted hereafter.

THE GEOLOGICAL RELATIONS OF THE PRINCIPAL NOVA SCOTIA MINERALS.

BY E. GILPIN, JR., HALIFAX, N. S.

(Ottawa Meeting, October, 1889.)

ANY estimate of the economic mineral value of an unvisited district is to the mining engineer largely a comparative one. If he knows that certain minerals characterize any given geological horizon he naturally draws upon his experience of the same ores as met by him under similar geological conditions. And if he has not had the personal experience, the investigation of the geology of a similar district as given in a trustworthy report enables him to lay a fair basis for conclusions.

In a general way these conclusions are of value, and while they pronounce on the possible mineral fecundity of a given district, they

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often give a decided rebuttal to startling statements of discoveries of ores.

The number of the geological horizons in Nova Scotia is limited, but they are well developed, in some cases typically, and exert a prominent effect on the agricultural and industrial distribution of its inhabitants.

The following table, based on Sir John William Dawson's *Acadian Geology*, will serve as an outline for my notes:

Modern.	
Triassic sandstone and trap.	
Permo-Carboniferous.	
Carboniferous	Upper coal measures.
	Productive coal measures
	Millstone grit.
	Marine limestone.
	Lower Carboniferous.
Devonian	Upper Helderberg.
	Oriskany.
Upper Silurian	Lower Helderberg.
	Clinton.
Lower Silurian.	
Cambro-Silurian.	
Cambrian, Longmynd series.	
Laurentian.	

This list is intended to refer rather to the divisions known to contain ores, than to be a complete geological scale of the province.

Geographically speaking, in Nova Scotia proper these divisions are represented as follows: On the Atlantic coast are the Cambrian of the gold-fields with their associated granites. Then follow the measures of the Cobequids running through Cumberland, Pictou, Colchester and Antigonishe counties, supporting Silurian and Devonian strata, the latter being met also in the western part of the province. The Carboniferous occupies the Gulf shore, and much of the district surrounding the Basin of Minas. The Triassic measures are met in narrow fringes and outliers around the Bay of Fundy. In Cape Breton the northern part of the island is largely composed of Laurentian, which occurs also between the arms of the Bras d'Or Lake. The Carboniferous rests on it, and forms fringes, while it is replaced in Richmond county by several large areas of Devonian.

Taking the divisions in descending order, their best known minerals are:

Modern.—Beds of bog iron-ore, manganese wad, peat, infusorial earth and clays are frequently met. A number of years ago the iron-ore was smelted in connection with older ores, at Clementsport

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near Annapolis. Deposits of considerable extent have been observed in Shelburne, Queens, Kings, and Pictou counties, and no doubt exist at many other localities. The wads are met in Cape Breton at several points, in deposits of considerable size, yielding from 20 to 40 per cent. of ore with water, iron, siliceous matter, etc. Smaller deposits are not unfrequently found, but as there is no demand they are not sought after. The deposits of infusorial earth have been used locally for insulating steam-pipes, and as an absorbent in the manufacture of dynamite.

Triassic.—The trap and associated ash-beds yield numerous varieties of zeolitic and other minerals. At several points they are penetrated by veins of massive and crystalline magnetite and specular ore of remarkable purity. The thickest vein that has come under the writer's notice, of magnetite, was about 15 inches wide.

At numerous points, most noticeably at Margaretville, copper-ores, principally carbonates with native copper, are found in veins in the trap and ash. These veins have been explored several times without success. No records have been made of the "low-grade" values of these rocks, and I believe from the frequent occurrence of copper-ores over so wide an extent of territory that, locally, beds may be found carrying the disseminated metal in amounts of economic value. The associated sandstones, red and friable, yield no building stone of value, and as yet have furnished to the miner only a few tons of manganiferous ochre.

Permo-Carboniferous.—In the great mass of sediments covering large areas in Pictou, Colchester, and Cumberland counties, and presenting fossil evidence of a transition from the Carboniferous to the Permian, there are few minerals of economic value. A few thin seams of coal are met, with fire-clays. At numerous points the sandstones and shales present irregular bedded masses and layers of copper-ores, principally gray sulphurets, with films and coatings of carbonate. These ores are associated with fossil plants to whose presence their deposition is to be attributed. Hitherto, attempts to find these ores in workable amounts have not been successful. A sample from Carribou, near Pictou, gave:*

	Per cent.
Copper,	40.00
Iron,	11.06
Cobalt,	2.10
Manganese,	0.50
Sulphur,	25.42
Lime,	0.92

* *Acadian Geology.*

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Admirable varieties of building stone, variously tinted sandstones and freestones have been quarried from these measures.

Upper Coal Measures.—These strata in general resemble those noticed in the preceding section. There is however more coal; and layers of clay iron-stone are frequently met.

Productive or Middle Coal Measures.—The principal mineral of this formation is coal, which is worked in Cape Breton, Pictou, and Cumberland counties. Deposits are known in five other counties, but have not been worked to any extent. The production last year was about 1,700,000 tons. The coal is bituminous, and the writer is not aware of the occurrence of any hard coal, although several discoveries of graphitic shale have been heralded as anthracite mines. Speaking in general terms the Cape Breton coal is the most bituminous, that of Cumberland less so, while the Pictou coals rather approach the semi-bituminous or steam variety. All the districts furnish coke of fair quality.

The following averages of analyses, from a paper on Canadian coals, read before the Montreal meeting of the British Association, will serve to show the variation in quality:

	Cape Breton.	Pictou.	Cumberland.
Moisture,	0.75	1.19	1.46
Volatile combustible matter,	37.26	29.10	33.69
Fixed carbon,	58.74	60.63	59.35
Ash,	3.25	9.34	5.50

The fire-clays accompanying these coals have never been systematically tested as to their value for fire-brick. Some that have come under my notice are too high in alkali and iron, while others theoretically were of satisfactory composition. An investigation into the capabilities of the fire-clays of Nova Scotia would form an object worthy the attention of the Canadian Geological Survey. Beds of cannel and oil coal, and of bituminous shale, are frequently met, but are not at present of economic value.

Beds of clay iron-stone (in a few instances of the black-band variety) up to a thickness of 12 inches are numerous. The following analyses made by me are of samples from the Pictou coal-field.

	Clay Iron-stone.	Black Band.
Moisture,	2.132	0.732
Iron protoxide,	45.361	36.000
Alumina,	16.962	3.180
Siliceous residue,	0.780	16.546
Lime,	trace.	3.780

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	Clay Iron-stone.	Black Band.
Magnesia,	1.655	0.783
Manganese,	trace.	4.450
Sulphur,	0.612	0.214
Phosphoric acid,	trace.	.586
Carbonaceous matter,	——	6.140
Carbonic acid,	——	27.589
Metallic iron,	35.00	28.000

Millstone Grit.—The dividing line between the millstone grit and productive measures is not a fixed one; and consequently there are workable beds of coal in its upper part arbitrarily referred to this horizon, but practically belonging to the productive measures. Throughout this range of measures thin seams of coal occur at numerous points, but in the presence of the larger and better known beds they have not yet received attention. At the Joggins, Cumberland county, large quantities of excellent grindstones are made for local use, and for export, and numerous beds of quarry rock are known.

Marine Limestone.—This horizon is very strongly developed in Nova Scotia, and to its presence is due the fertility of the central and northern part of the province, for its gypsum, limestone, and marl yield soils of permanent fertility. Its most prominent mineral is gypsum, in every variety and texture, which occurs in beds in many cases extending for miles, and reaching in thickness 100 feet. It is quarried for export to the United States, chiefly near Windsor, in the Basin of Minas, the annual production varying from 100,000 to 150,000 tons. A few thousand tons are annually sent up the Gulf of St. Lawrence from Cape Breton, but the export trade, on account of the difference in freight, is from the Windsor district. Limestone is equally abundant, and is burned for local use, and used in rough masonry, and at Walton, near Windsor, furnishes a beautiful red stone, which it is said will first be used in the new buildings of a prominent New York newspaper. At several points these limestones carry deposits of manganese, lead, iron, and copper-ores and barytes. The manganese is met as pyrolusite with a little hard ore, and is of remarkable purity, carrying very minute amounts of iron. It brings a price varying up to $100 a ton, but the demand is limited. I am not aware of any deposits of these ores adapted to the steel-makers' purposes. Tenny Cape, Onslow, and Loch Lomond are the best known mines.

The iron-ores are limonite, red hematite, and spathic ore. As yet the ores of this series are little worked, but they are of undoubted

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value. The purity of some of them may be shown by the following analysis of a sample from Brookfield:

	Per cent.
Water,	11.36
Silica,	1.54
Sulphuric acid,	trace.
Phosphoric acid,	trace.
Metallic iron,	60.00

The spathic ore occurs in beds, the limonite and red hematites as contact and replacement deposits. The lead-ores occur as small segregated veins, and as aggregates. The silver contents of the Nova Scotia lead-ores are variable; assays have returned as high as 100 ounces, but the average would not exceed 10 ounces. The copper-ores resemble in composition and mode of occurrence those already described. Celestine, fluorspar, and brine and mineral springs also occur in these measures.

Devonian (Upper Helderberg, Oriskany).—In the hills lying to the south of the Annapolis valley are numerous important bedded deposits of magnetite and hematite. The ores vary in character, but some are of excellent quality and favorably situated for mining and exportation. In Guysboro' county the Devonian measures carry several large deposits of specular ore of good quality, but as yet unproven. The ore is presented in veins and in large masses, and may be connected with the dioritic dykes found at this point. It occurs under similar conditions near St. Peters. I believe that by some the copper-ores of Polsons Lake and Lochaber in Antigonishe county are referred to this horizon. These deposits are connected with igneous dykes, and, as far as they have been explored, appear very promising, but their distance from shipping, etc., has diverted attention from them.

Upper Silurian (Lower Helderberg, Clinton).—In Pictou and Antigonishe counties, strata which are referred to this age carry important deposits of bedded red hematites, varying in thickness up to 50 feet. The ores are siliceous, but frequently free from sulphur and phosphorus; their metallic contents vary from 35 to 50 per cent. Owing to their size and accessibility they can be cheaply mined, and will probably be utilized in connection with the richer ores found in their vicinity.

Lower Silurian.—These measures as developed in Cape Breton carry several beds of red hematite, and deposits of copper pyrites and sometimes cupriferous iron pyrites.

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Cambro-Silurian.—This term has been provisionally applied to the mass of strata forming the Cobequid hills and reappearing in Pictou county. The ores occurring in these measures embrace several varieties of hydrated and anhydrous peroxide. At Londonderry the limonite occurs with a little specular in a large interstratified vein containing ankerite and sideroplesite, the latter being used to some extent in the furnaces of the company operating at this point. Magnetites have also been met in parts of this range. In Pictou county the ores occur in interstratified veins with ankerite, but are principally specular. Copper- and gold-ores are reported as occurring in this range, but I am uninformed concerning their economic value. The deposits of antimony-ore at Rawdon, Hants county, have been worked for several years, and are probably extensive. Their exact geological horizon is not yet clearly known.

Cambrian (Longmynd).—The only mineral of economic value yet found in these measures is gold. The strata are slate and quartzite lying in large abrupt folds, which have permitted the intercalation of the gold-bearing quartz-veins, varying in thickness up to 12 feet. The gold is also found disseminated in some of the beds of slate. Copper, lead, zinc, iron and molybdenum sulphides occur with the gold in the quartz-veins, but not in amounts of economic value. The auriferous territory is estimated to cover 3000 square miles, and as the annual production of gold is only about 23,000 ounces, there is an ample opening for miners.

Laurentian.—This series is widely distributed in Cape Breton, and roughly speaking is divisible into the felsite and the limestone series. They contain red hematite, in places magnetite, of excellent quality, although some of the ores are reported to be rather high in phosphorus for Bessemer purposes. Copper-ores, graphite, asbestos and mica are also found. The limestones are frequently altered into marbles. Some of the deposits are beautifully tinted, and are said to be adapted for building and statuary purposes. At present the West Bay (Cape Breton) marble is largely burned into a lime of excellent quality, which is used in the province, and exported to the United States, etc. To complete the parallel between these Cape Breton strata and the Laurentian rocks of the vicinity of Ottawa, there remains to be discovered in the former "Phosphate Rock," and there appears to be no reason for its absence.

Copper ores are found at numerous points and are undoubtedly valuable as they are widespread and at the surface show well, but up to this date no decided steps for development have been made

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except in the case of the copper-ores of Coxheath, near Sydney. Here the Eastern Development Company have proved several large veins carrying copper in workable amounts to a depth of 300 feet. The ore is chalcopyrite with erubescite, and carries considerable amounts of gold and silver. Molybdenite also occurs in small disseminated grains and nodules at several points, and a few lots have been shipped to England.

In summing up these brief notes it may be said that the visitor to the province would place the minerals, irrespective of any development effected, in the following order: Coal, Iron, Gold. It is perhaps unusual to find these three so close together.

NOTES ON THE REPUBLIC OF COLOMBIA, S. A.

BY JOHN C. F. RANDOLPH, E.M.

(Recently National Commissioner of Mines for the Department of Tolima.)

(Ottawa Meeting, October, 1889.)

NEW GRANADA, or, as it is known to its people, the Republic of Colombia, occupies the northwest corner of South America, and includes the Isthmus of Panama. Although it is one of our nearest neighbors, so little is known about it in the United States that the following notes regarding it, flowing from a year's professional residence, may be timely. It is bounded on the north by the Caribbean Sea and Costa Rica, on the east by Venezuela and Brazil, on the south by Brazil and Ecuador, and on the west by the Pacific Ocean. Its extreme geographical limits are from 2° South latitude to 12° North latitude, and from 69° to 78° East longitude. In size, therefore, it is roughly 1000 miles from north to south, and 600 miles from east to west.

After many political convulsions and revolutions, and a somewhat checkered history, the country consists to-day of nine departments, Panama, Bolivar, Magdalena, Cauca, Santander, Antioquia, Tolima, Boyacá and Cundinamarca. These departments are governed from the national capital of Bogotá, although each has its own capital and departmental staff. The total population of the Republic is not more than 3,000,000 inhabitants, and these to a large extent consist of mixed races. Those who live in the hot belt are, as a

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rule, indisposed to exertion and indolent, while those living in the mountains are more energetic. The inhabitants of Antioquia are esteemed the most enterprising race in the country. They are a mountain people, claiming direct descent from a colony of Moorish Jews which came early from Spain, and even when they emigrate to other states they always seek altitudes above 4000 feet in which to live. The Antioquians, as a rule, make the keenest politicians, the bravest soldiers and the shrewdest traders. The government is an unsettled one, subject to frequent revolts and revolutions. The governmental methods are usually arbitrary, the action of the courts is uncertain, the currency depreciated, and domestic and foreign credits precarious.

Topography.—The great chain of the Andes, entering Colombia from the south, splits up into three well-defined mountain ranges, known respectively as the Cordillera Oriental, the Cordillera Central, and the Cordillera Occidental. These ranges seldom exceed 8000 feet in height, although they contain a number of snow-clad peaks like the Pic of Tolima and the Nevados of Huila, Herveo, Santa Ysabel and Santa Marta, all of which attain an altitude of at least 16,000 feet. A much greater altitude has been claimed for the Pic of Tolima, but no measurement has ever been made. The flanks of these ranges are extremely precipitous and rugged, and have been cut by the active forces of disintegration and erosion, into great ravines, cañons and gorges, separated by knife-edged cordons or ribs falling away from the main ranges. In probably no country in the world have the natural forces of rain and heat been such important factors in moulding the topography. There are no roads in the country, but only narrow bridle-paths. This extreme asperity of surface makes the country a most difficult one for mule transportation.

The valley of the Magdalena, between the Eastern and Central Cordilleras, contains the main water-way of the country, and the larger part of the population is found in the adjoining departments. In point of size and volume of water, the Magdalena River compares best with the Ohio River in North America. It is, however, similar to the Mississippi in its tendency to form new channels, and in forming a delta where it debouches into the sea. It rises in the mountains near the Equator, and flows north for 900 miles to the Caribbean Sea. For the lower 200 leagues of its tortuous course the river is navigable at all seasons as far south as Honda. A rapid divides the navigation of the lower river from that of the upper

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river at Honda. From Honda to Purificacion, a distance of 95 miles, the upper river is navigable by smaller steamers, and still smaller ones are used in ascending from Purificacion to Neiva, a further distance of 60 miles. The fall from Neiva to Purificacion is 300 feet, from Purificacion to Honda 250 feet, and from Honda to the Caribbean Sea 650 feet. All the boats used are flat boats, with wheels at the stern. In the dry season the upper river is not navigable for lack of water and the lower river is so very shoal, that many accidents occur to the steamers. The second great valley of Colombia lies between the Central and Western Cordilleras, and contains the Cauca River, which is the principal affluent of the Magdalena.

Seasons.—The seasons of the Republic of Colombia consist of two rainy and two dry seasons each year. The former begin ordinarily with the equinoxes, and the latter with the solstices. The normal length of each season is three months, but on the coast the wet seasons are the longest, while far in the interior the dry seasons are frequently the longest. Abnormal years are not infrequent, in which a dry or a wet season may continue six or seven months without intermission. In the wet season, there occur only one or two short rains each day, in which an enormous amount of water falls, making the trails almost impassable and causing an exuberant vegetation.

Range of Temperature, etc.—The peculiar topography of Colombia, consisting of great mountain ranges alternating with broad river valleys, gives rise to a great diversity of temperature and of climate, not ordinarily considered possible for equatorial regions. The daily range of temperature depends solely on altitude. The belt of tropical temperatures includes the river valleys and all altitudes below 3000 feet. At 3000 feet is a perennial Italian climate of eternal June, when the range is 85° F. by day and 80° at night. At 4000 feet the day temperature is 75° and the night temperature 70°. At 5000 feet the temperature by day is 70° and 65° by night. Finally, at 8300 feet, the altitude of Bogotá, the day and night temperatures are respectively 60° and 50°. These ranges of temperature due to altitude are perennial, and independent of the seasons, and if the altitude of any given locality in the Republic is known, its climate can be inferred with unerring exactness. The true equator of temperature passes through Barranquilla and Colon, although they are 10° north of the geographical equator. The following list shows the observed altitudes and daily ranges of temperature in the principal towns of Colombia:

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Barranquilla, sea level,	95° F., upward.
Aspinwall (Colon), sea level,	95° " "
Honda, 650 feet altitude,	95° day, 90° night.
Neiva, 1200 feet altitude,	90° " 90° "
Mariquita, 1800 feet altitude,	90° " 85° "
El Hato, 2500 feet altitude,	90° " 85° "
Altamira, 3000 feet altitude,	85° " 80° "
Bucaramanga, 3000 feet altitude,	85° " 80° "
Cali, 3500 feet altitude,	85° " 80° "
La Mesa, 4000 feet altitude,	75° " 70° "
Socorro, 4000 feet altitude,	75° " 70° "
Ibagué, 4300 feet altitude,	75° " 70° "
Medellin, 4900 feet altitude,	73° " 65° "
Popayan, 5571 feet altitude,	70° " 60° "
Bogotá, 8350 feet altitude,	60° " 50° "

Because of this great diversity of climate the country is far more healthful than most equatorial regions.

Yellow fever always exists on the coast and is occasionally epidemic there, but in the interior of the country only occasional sporadic cases occur at altitudes below 1500 feet. Bilious and intermittent fevers, dysenteries and diarrhœas are, however, prevalent, due to improper food and exposure.

The soil of Colombia is fertile and produces dye-stuffs, gums, sugar, tobacco, quinine bark, india-rubber, coffee, chocolate, and cereals. Only human industry and cheap transportation are lacking. The soil is so rich that the same land will yield three crops in one year to very slight labor, but this labor is seldom applied. Infant railroads have been begun in several places, but lack money both for construction and maintenance. Only one of them is as long as thirty miles.

Geology.—The geology of Colombia is simple and in many respects similar to that on the western side of our own continent; while only the geology of the central departments of Tolima and Cundinamarca can be stated from personal experience, there are good reasons for believing that the same geological features extend both northward and westward. The department of Tolima lies mainly on the west side of the Magdalena, on the eastern slopes of the Central Cordillera, and on the east side of the river the department of Cundinamarca rises upward into the Eastern Cordillera. The valley of the Magdalena is a synclinal valley and the country rises rapidly away from the river on both sides. On the Tolima side broad llanos of alluvion are first traversed which are cut through by foothills of red Triassic sandstone, which present abrupt broken cliffs to the west,

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and gentle slopes to the east. At a height between 4000 and 5000 feet above the sea, a broad zone of Archæan schist is encountered, consisting at various points of micaceous, chloritic, talcose, hornblendic or siliceous varieties. These schists all coincide in strike with that of the range and dip to the east. They are the oldest rocks known in Tolima, and are probably of Laurentian or Huronian age. They have been exposed by the breaking of the overlying Triassic formation at the time of the upheaval of the central range, and are cut through continually by dikes of andesite and other eruptive rocks.

This belt is found as far west as the flanks of the Paramo of Ruiz at 6000 feet altitude, and although exploration has never as yet been possible further west, evidence is not lacking that at many points among the higher peaks of the range, the eruptive rocks have broken through the schists, and form the backbone of the Cordillera. At some few points the Triassic sandstones are overlaid with thin patches of Jurassic limestone containing characteristic ammonite fossils. It is therefore probable that the elevation of the Central Cordillera took place either at the end of the Triassic or at the beginning of Jurassic time. This was evidently an age of great movement on both continents. Subsequently occurred the action of the great forces of erosion in the Quaternary, producing the alluvion, and gold-bearing gravels of Tolima. The eras presented, therefore, in the geological history of Tolima, are: Archæan, Jurassic, Triassic and Quaternary. The same geological features are attested by competent observers, as extending northward through the state of Antioquia and are found on the west of the range. In the southern part of Tolima the range subsides into well rounded hills, attaining a maximum altitude of 7000 to 8000 feet, and the Triassic sandstones are unfractured.

On the east side of the river, the Eastern Cordillera rises rapidly to Bogotá. The same geology is encountered, except that the Jurassic limestones are found of very great thickness overlying the Triassic sandstones. There is no evidence in Cundinamarca that in the upheaval of the Eastern Cordillera, the sedimentary rocks were fractured, but it is not impossible that in the northern states of Boyacá and Santander the schist formation may be encountered, due to such a cause. There is some evidence to show that the upheaval of the eastern range took place at the end of the Jurassic and slightly subsequent to that of the Central Cordillera.

In all the area described, which is equal to one-fourth of the Re-

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public of Colombia, the Palæozoic rocks are not encountered. In the Triassic formation abundant lean lignites and black asphaltic oils are found, as well as some evidences of copper. In the black Jurassic limestone of Boyacá are the celebrated emerald mines of Musa. In that division of the Archæan schists which consists of mica schist, veins are occasionally found containing free gold at Ibagué, Bermillion and Aneime, and in the silico-chloritic class of schists, veins carrying ruby and black silver. Galena is notably rare in Tolima and other departments of Colombia.

As to the geology of the third or Western Cordillera only the conjecture can be offered, that this range also dates its elevation to some period during Jurassic or Triassic time. The valley of the Cauca river may be a second Triassic synclinal valley, but this impression rests solely on answers to questions asked of those familiar with the Western Cordillera and not on personal observation.

Mines and Mining.—The principal mineral belt thus far discovered in Colombia is on the eastern slopes of the Central Cordillera in the departments of Tolima and Antioquia. The gold gravels of this belt have hitherto been more profitable than the metalliferous veins. Even in ancient times gold-washing was pursued successfully, as is evidenced by the gold ornaments to-day dug from the "guachos" or ancient Indian graves. The larger part of the gold gravels of Tolima are in small thin patches, which are intermittently worked by the natives by the "*barèque*" (a crude method of ground sluicing) and by the "*batea*," and afford in the aggregate a considerable amount of gold per year. The few larger gravels, of sufficient importance and value for persistent effort and work in the large way, are in the hands of foreign companies. As a rule it may be said that none of these gravels are more than 50 to 70 feet in thickness, that they are all troubled with pipe-clay, and that their return per cubic yard is less than the gravels of California. It may be also stated that the working year is much curtailed by lack of water in the dry season, and by the bursting of the ditches in the rainy seasons. Two successful English companies, the "Malpaso" and the "Oritá" on adjoining areas of gravel 75 feet thick have been profitable. The "Malpaso" operates three monitors with 1000 miner's inches of water, under a head of 125 feet, and pays monthly $15,000 to $20,000. The "Malpaso" is underlaid by 3 feet of rich cement. The "Oritá" operates two monitors with 800 miner's inches of water under a head of 100 feet, and pays monthly $5000 to $7000. Both properties are at times greatly troubled with pipe-clay. An American company operates profitably

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at Charcas on the Saldaña river on gravels 60 feet thick, but lacks sufficient water. Other good gravels exist on the Saldaña river, but the obtaining of water in sufficient quantities and pressure will be difficult. A new English company is making a ditch 15 miles long which will take 3000 miner's inches of water from the river Guali under a head of 300 feet to work the "La Rica" property which has already been satisfactorily tested. An American company known as the "El Talento" owns the "Cájongra" and the "Boconá" in the same range as the "Malpaso" and "Oritá," but has as yet expended no money on the properties. Another American company working the "El Campeon," was forced to close down on account of great thickness of pipe-clay and very thin gravels, and an English company testing the gravels of "Pompona" did not find either sufficient gravels or water. It is not known that hydraulic properties are worked in Antioquia with water under pressure, but small spasmodic enterprises are abundant among the natives. Several foreign enterprises are reported in the Choco valley and elsewhere in the republic; but no notable results have been obtained.

The veins of Tolima are veins of quartz intercalated in the schists. These veins have a general tendency to wedge out and reappear in sympathy with the mobile curves of the schists. No true vein, cutting the formation, has as yet been found in Tolima. Thousands of veins of slight importance have been denounced in Tolima solely for the purpose of a possible foreign sale, although in very few cases have even prospect holes been put upon them. As a rule they are of little value and very thin. A few gold mines have been worked at Ibagué, Libano and China by native companies without profitable results. The formation of mica schists offers the hope that more careful explorations will bring to light gold-bearing veins, cutting the formation, and of more assured length, depth and value of ore. This is especially true of Anaimé and Bermillion, and of a large unprospected area of mica schists west of Ibagué. The native or Antioquian stamp-mill costs very little, is operated by water, stamps 500 pounds per head in the 24 hours, has no mortar, and is rather inferior to the stamp-mills figured by Agricola, in "*De Re Metallica*," three centuries ago.

A number of silver-mines are to-day worked in the belt of chlorito-siliceous schists, in which the ores, when produced, contain ruby and black silver. These ores can not be locally treated on account of the high price of salt. Hand-picking and concentration are therefore resorted to, and transportation to America or England

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is necessary. Of these the Frias mine, owned by an English company, has been at times very productive, and to-day has ore on the 540-foot level. All the ore in this mine has been brought in by the intersection of a cross seam with a main vein. The ore is first hand-picked up to 250 ounces, and all lower grades are concentrated on jigs and buddles to the same grade. All sortings and concentrations are shipped to England, at a cost of $65 per ton, and are subject there to smelters' and agents' charges. It is estimated that the expenses of mining, concentrating, shipping and reducing, consume fully 100 to 125 ounces of any value per ton contained. The "El Cristo" mine is an American property, situated in the midst of and parallel to veins in the schist that are known to continually pinch out. It is not open to inspection. As yet but little ore has been produced, or shipped by it, although it has been worked since 1885. The ore carries ruby and black silver. Two English ventures, denominated the "Calamonte" and the "Santa Maria," lie between the "Frias mine" and the "El Cristo" in the same schists. Although vigorously worked for several years, and extravagantly equipped with concentrating mills and compressed-air drills, they have not as yet produced sufficient ore to justify hopes for the future, and their funds are almost exhausted. The same statement is true of the "Silencio" and "Organos" mines at the head of the Saldaña River, both of them English enterprises. A new American mine has been opened at "Bocanéme," north of "El Cristo," which, by striking a considerable body of ruby silver-ore of good grade at 180 feet in depth, excites very lively hopes for the future. The vein is located on the edge of the schist formation, and near a dike of andesite, and may prove very profitable, but its ore will be subject to heavy charges for shipping to America.

Near the "El Cristo," in the same district, and intercalated in the same schists, is the abandoned silver-mine known as "Santa Ana," which has always been esteemed the greatest silver-mine ever worked in Tolima. It was worked continuously from 1826 to 1878 by an English company, and when abandoned was 750 feet in depth. After 52 years of persistent work it was abandoned because its expenses had exceeded its receipts by more than $1,000,000, and during its history it had enjoyed but two short periods when it was worked at a profit.

These are *all* the vein-mines worked in Tolima in 1888 that are worthy of record here, on account of the effort and intelligence applied to them.

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The enterprises in mining in Antioquia have also been very few indeed, and a very uncertain product. The most profitable mine worked in Antioquia is said to be the "Zancudo," of which most extravagant stories are told without authenticated details. Very few mines are worked in Cauca and other parts of the Republic, but the number of unprospected mining claims is very great.

As yet no gold or silver veins of marked value and product, as judged by North American standards, have been discovered in Colombia, and very few foreign investments have been remunerative. The history of the Santa Ana mine excites distrust of veins included in the schist in this locality. The best chance for the mining industry of Tolima, and perhaps of all Colombia, depends on the discovery of gold and silver veins, cutting the schist formation, and not included in it, and especially of veins near the eruptive rocks breaking through the schists.

A very large annual gold product has always been claimed for the Republic of Colombia, without any authentic figures being given to sustain the claim. It is not impossible that as much as $2,000,000 annually may be produced, mainly coming from very small enterprises in ground-sluicing thin patches of gravel of slight area. Much territory in Colombia still remains to be explored, undoubtedly, and valuable mines may be actually discovered; but the Republic of Colombia does not to-day offer an attractive, profitable, or safe field for foreign investment, and statements as to rich mines in that country should be received with more than ordinary caution.

BIOGRAPHICAL NOTICE OF WILLIAM H. SCRANTON.

BY R. W. RAYMOND, NEW YORK CITY.

(Ottawa Meeting, October, 1889.)

In the death of William H. Scranton, which occurred at Oxford, N. J., June 19, 1889, the Institute has been called once more to mourn the loss of a name from the fast diminishing list of those who constituted its membership in the first year of its existence. But it loses more than a name—a friendly presence and an intellectual force have departed; a fruitful life has been cut short.

Mr. Scranton was born in Belvidere, N.J., on the 13th of January, 1840. He was the eldest son of Colonel George W. Scranton, the

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founder of Scranton, Pa., and in the critical years of 1860 and 1861 the representative in Congress of the Scranton district. The family removed from Belvidere to Scranton in 1840, while William was an infant. After five years of schooling at Montrose and Wyoming, Pa., and Stamford, Conn., he entered the Rensselaer Polytechnic Institute in 1857. Leaving the institution about the end of 1860, he settled himself at Oxford, N. J., as civil and mining engineer for the firm of G. W. and S. T. Scranton, which became, in 1863, the Oxford Iron Company. Of this company he continued the engineer up to 1873, constructing during this period the buildings, furnaces, etc., which still stand at Oxford. In 1873 he became general manager of this company, taking charge of its mines, works and business affairs, and the position he retained after the concern was reorganized as the Oxford Iron and Nail Company in 1878, and until he resigned his office in November, 1885. In January, 1886, he became general manager of the Fall River Iron Works in Massachusetts, and upon the discontinuance of that enterprise a year or two later, wound up its business and returned to his old home at Oxford, where he occupied himself with his practice as a consulting engineer, and with the pursuit of professional investigations and commercial undertakings on his own account. One of the latest of these was the introduction into the United States of the Wenström magnetic separator, recently described in the papers of Mr. Cook and Mr. Birkinbine in Volume XVII. of the *Transactions* of this Institute. Both the merits of this machine and the field of its practical usefulness were directly connected with the favorite study of his life, to which I will presently recur.

The career of an engineer and general manager, remaining for twenty-two years at one place, must be largely one of routine, and its achievements are too often unappreciated for lack of an adequate record and a competent historian to interpret it and call attention to it. When a man thus engaged makes his own record in professional papers, his colleagues, and ultimately the public, come to know what he is doing, to recognize his ability and to applaud his success. But if he is too busy or too bashful to give to others this chance of becoming acquainted with him, he may receive far less praise than he deserves. Both these hindrances existed in the case of Mr. Scranton; and a third may be added, namely, the circumstance that he began his professional work at a time when there was, in this country, no such wide and accessible professional public, and no such organized means of general intercommunication among its members, as the rapid

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development of our technical journals and technical societies has furnished since. That he was quick to recognize the advantages of such intercourse, his prompt adhesion to this Institute, and the unfailing interest which he manifested in its proceedings, abundantly testify. But he could not so suddenly form the habit of a free utterance of his own experience and opinions, and by his unconquerable timidity and reticence he lost much, while his colleagues lost yet more. It is therefore, with scanty materials, drawn partly from my own reminiscences, partly from the brief memoranda of personal friends, that I construct a tribute to his work and character less complete than they deserve.

Mr. Scranton was an accomplished draughtsman, designer, and architect. The large blast-furnace at Oxford, of which he made the plans and superintended the construction in 1872, and which he remodeled and improved in 1855, is a standing evidence of his skill. In its day it was a model of thorough and advanced scientific construction, and it has long had the reputation of producing excellent pig-iron at exceptionally low cost. As all furnacemen know, the cheapness of raw materials and the reduction of labor-cost in unnecessary handling are important elements of such a result; but this fact does not diminish the credit due to Mr. Scranton, for the mining and transportation, as well as the general metallurgical and mechanical direction, were under his charge and received the impress of his mind.

As a manager of workmen his success was extraordinary. Employing from 800 to 1000 miners, furnacemen, mechanics and laborers of all classes, he commanded so entirely their confidence and respect that he never encountered a strike. Both capitalists and workingmen recognized his strict justice and absolute integrity, as well as the kindness of his disposition, and men of all classes became and remained his friends.

He was a constant student of books, making copious notes of his reading, as well as his original experiments and investigations. His note-books* were models of fulness, order and critical analysis. His house at Oxford was the home of domestic happiness and refined and cordial hospitality; and those who enjoyed the privilege of tarrying beneath its roof found in their host a surprising range of knowledge,

* An invaluable collection of these note-books, with drawings, calculations, etc., has disappeared since his death, through the dastardly outrage of the robbery of his private office by unknown parties.

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and a power of imparting it which his habitual reserve in public did not indicate.

The principal contribution which Mr. Scranton made to the practice of our profession (though, unfortunately, not in satisfactory measure to its literature) was connected with the use of the magnetic needle in the survey of iron-ore deposits. His interest in this subject began at a very early period. His mother's brother, Mr. James Hiles, of Belvidere, N. J., a land surveyor by profession, was a man of conscientious accuracy, original force and sympathetic temperament, highly esteemed as an authority on engineering matters by the people of that region. Doubtless the influence of this uncle, with whom young Scranton used to spend much of his time, and who delighted to encourage and guide the boy's ambition, had much to do with his choice of a profession. Mr. Scranton was accustomed to speak of his uncle with much admiration and affection, and to acknowledge his own indebtedness to this happy early companionship for inestimable training and inspiration. When Mr. Hiles died, about twenty years ago, his maps and papers were left to the nephew, who had been his pupil.

Mr. Hiles had satisfied himself of the inaccurate and misleading character of the indications of the dip-needle as ordinarily used by miners, and had developed by himself a system of magnetic surveying calculated to reduce the sources of error and to increase greatly the definiteness of interpretation. He had constructed several dip-needles of improved form, two of which (among the first, if not the very first, made in this country) are still preserved in the house at Oxford, The Scrantons being heavily interested in iron-mines and iron-ore bearing lands, William Scranton was, from the beginning, called upon to make numerous surveys of such properties and devise plans for their development. This led him to perfect the system conceived by his uncle, and the maps of magnetic surveys which he constructed twenty-five years ago prove him to have been one of the first to develop the systematic determination and interpretation of magnetic intensities, as well as dips for tracing the limits of underground deposits of magnetite. His practical skill in the use of his system was extraordinary, and it should be noted that his conclusions were over and over again submitted to the severe test of immediate trial in mining. His surveys were made the basis for the location of shafts, the running of drifts and levels, the purchase of land, and the calculation of necessary ore-supplies. This test they stood so well as to give to him great confidence in the method, and to others

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great confidence in him. But he was always anxious that Mr. Hiles, as the real originator of the system, should receive the credit for it.

Dr. George H. Cook, the State Geologist of New Jersey—another honored member of the Institute whose still more recent death we mourn—recognized the ability and authority of Mr. Scranton in this field, and the report of the State Survey for 1879 contains a chapter in which the system of Scranton is outlined, though not fully set forth. The reason of the delay in the public description of a method already used for years is curiously characteristic of Mr. Scranton. Early in his connection with the Institute as a member, he had determined to prepare a paper on the subject, both as a vindication of the memory and merit of his uncle, and as a useful contribution to an important department of mining practice. Once, at least, he had come to a meeting of the Institute with such a paper in his pocket, and had carried it home again unread, because he was too bashful, when the critical moment came, to get up and read it. Angry with himself for such timidity, and resolved to overcome it at all cost, he made another effort, and once more (in February, 1876, I believe) brought the paper to a session of the Institute. Unfortunately, while he sat fighting within himself for courage to face even that harmless audience, a fellow-member, Prof. J. C. Smock, of the New Jersey Geological Survey, presented a paper on the same subject. Prof. Smock in New Jersey, and Mr. T. B. Brooks in Michigan, working on the same lines, had arrived at something like the system of Mr. Hiles. If Mr. Scranton had had a little less modesty or a little more experience in such matters, he would have known that the presentation of Prof. Smock's paper rendered his own the more appropriate. But, to his fancy, it took away his last chance of utterance, and once more he silently carried his manuscript home.

In later years he overcame, to some extent, this reluctance to speak his views. At least he contributed valuable papers and maps to the New Jersey reports. In 1888, and down to the time of his death, I believe, he was regularly connected with the State Survey. Dr. Cook imported from Sweden a modern magnetometer, the product of the latest improvements made by Swedish skill in the art of magnetic surveying. Mr. Scranton recognized at once its superiority to the dip-needle, and, in association with Mr. Uno Sebemus, a Swedish engineer, sent over to expound the merits and show the application of the new instrument, he operated with it in field-work during the

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last year of his life, made improvements upon it, and incorporated it in the system which he had perfected.

Once more (and this time without risk of being frightened from his intention) he determined to present to the Institute the ripe results of his study and practice, and at the time of his death he had arranged the materials for a paper on the magnetic survey of iron-mines, and had prepared portions of the paper which was, unfortunately, among those stolen. This would have been probably the best comprehensive treatise on the subject, combining foreign with American progress, and adding much valuable criticism and suggestion.

Upon the monument of his father, in the family plot at Dunmore Cemetery, at Scranton, are engraved these words:

"Kind-hearted and benevolent, genial and true in his relations with his fellow-men; a man of noble purposes and high Christian character; he was called to his reward in the midst of his usefulness, loved and mourned by all who knew him."

The worthy son of such a father sleeps now beside him, and the same inscription may fitly stand for both.

BIOGRAPHICAL NOTICE OF GEORGE H. COOK.

BY JOHN C. SMOCK, ALBANY, N. Y.

(Ottawa Meeting, October, 1889.)

GEORGE HAMMELL COOK was born at Hanover, Morris County, New Jersey, January 5, 1818. His early education was obtained in the country school, and he was, for a short time, a teacher in his native town. In 1836 he was engaged, as a civil engineer, in making surveys and locating the line of the Morris and Essex Railroad from Newark to Morristown, and the extension of the road to the Delaware River. Appreciating the advantages of a thorough education, and seeking a wider field of usefulness, he entered the Rensselaer Polytechnic Institute at Troy in 1838, and was graduated from that school in 1839. In this institution he heard the lectures of Amos Eaton, the famous teacher, from whom so many of our older scientists caught their inspiration and love for the natural sciences. After graduation, he was employed as a tutor in the school, and subse-

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quently as adjunct professor. In 1842 he became senior professor, giving instruction in the natural sciences. Here he remained until 1846, attracting students and enlisting the support of influential and wealthy friends in behalf of his *alma mater;* and throughout all the years of his busy life he showed the same steady interest in its prosperity. In 1846 he went to Albany, where he was engaged in manufacturing business for a short time, until he was called to the chair of mathematics and natural philosophy in the Albany Academy. He was appointed principal in 1851. While in Albany he was chemist of the Onondaga Salt Springs, and made four annual reports on them. He made his first visit to Europe in 1852, for the purpose of studying the salt manufacture of England and France. In 1853, in response to a call from Rutgers College, he returned to his native State, and accepted the professorship of chemistry and natural philosophy. For thirteen years he taught botany, mineralogy, chemistry, and geology, giving all the instruction in the natural sciences. The college museum was founded by him, and its collections grew rapidly through his untiring efforts and the aid of friends whom he enlisted in its support. He continued as professor, and later as vice-president of the college, up to the time of his death.

The subject of a better education of the farmer, and the claims of agriculture upon science, pressed upon him; and it was largely due to his patient and persevering efforts, and his wide influence among practical men, that the Morrill bill, creating and endowing colleges for the benefit of agriculture and the mechanic arts in the several States, was passed in 1864. Rutgers Scientific School was organized in 1863, and the State College in 1865, through his earnest and successful appeals in behalf of the scientific training of the industrial classes and the advantages which were offered at New Brunswick. The increased teaching force, necessitated by the newly-founded school of science, afforded him some relief in his college work, and his professorship was changed accordingly. From 1880 to his death it included geology and agriculture—his favorite subjects of study. He was vice-president of the college from 1864 onward—a quarter of a century. His loyalty to Rutgers was remarkable, and unsurpassed by that of any of her own graduates; and he has left a monument to his well-directed labor in its enlarged prosperity as well as in the hearts of the many students who were under his care.

Dr. Cook's work in the geological survey of New Jersey began in 1854, in which year he was appointed assistant geologist in charge

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of the geological survey of the southern division of the State, and was associated with Dr. William Kitchell, State geologist. His reports on the green-sand marl-beds, on the changes in the coast-line and the subsidence of the shore, and the final report on the county of Cape May, give the results of his survey of that part of New Jersey. The most notable of these publications was that in which he described the structure of the green-sand marl-beds. The subsidence of the coast was another important geological fact first observed by him. It was announced, in 1857, in a paper read at the meeting of the American Association for the Advancement of Science, held in Montreal. The geological survey was suspended from 1856 until 1864, when it was resumed by legislative enactment, under his direction as State geologist. The results of the work were published, in 1868, in the *Geology of New Jersey*. They were accepted with great favor, and the survey was ordered continued by an almost unanimous vote of the legislature. By further and supplementary acts, passed in 1872, 1876, 1880, and, lastly, in 1885, the geological survey has been continued. Annual reports have been published for the years 1863 to 1888, inclusive, excepting 1868. In 1878 the clay deposits of the State were described in a volume entitled: *Report on the Clay Deposits of Woodbridge, South Amboy, and Other Places in New Jersey, with their Uses for Fire-brick, Pottery, etc.*

The topographical survey had its origin in the local maps of this clay district, but the fully organized topographic staff was not put in the field until 1880. The completed results of that department have appeared in the accurate and beautiful *Atlas of Maps*, and in the volume descriptive of the topography. The value of these topographic maps to the civil and mining engineering professions is proven in the use to which they have been put and the large demand for them.

The plan for the completion of the geological survey included catalogues of the minerals and mineral localities of the State, of the plants, and lists of the species belonging to its fauna. A general summary of the results of the geological investigations, with reports on the palæontology, were to fill up the well-matured and comprehensive plan of this great work of his life. One volume, devoted to the geodesy, topography, magnetism, and climate, had been published; the second volume of the final report was nearly ready for the press at the time of his death.

The agricultural interests of the State were as dear to Dr. Cook as its geology, and the farmers were his warm friends and stanch

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supporters in all of his plans to advance the cause of agriculture. The State Board of Agriculture was organized at his suggestion, and he was its efficient secretary until 1879. The New Jersey Agricultural Experiment Station, established in 1880, was the consummation of a long-cherished plan for the promotion of agriculture; and after eight years of successful work he lived to see the State acknowledge the value of the station by erecting, at New Brunswick, New Jersey Hall, a building devoted to scientific investigations and experiments in this wide field.

Another State institution which owes to him its origin is the New Jersey State Weather Service, started in 1886. He was its director.

Dr. Cook was an active member of the State Board of Health, and served on its executive committee. His intimate knowledge of the State gave value to his counsels in the cause of public health, and he was particularly interested and active in urging the needs of wholesome and abundant water supply for the cities of the State.

For several years he had been one of the proprietors of East Jersey, and the surveyor-general of the board.

Elected in 1874 a member of this Institute, he felt a strong common bond of attachment to men engaged in the practical development of the natural resources of the country. His early education and work as an engineer, and his official duties as State geologist enabled him to appreciate the services of the profession in developing our mineral wealth. His paper "On the Southern Limit of the Last Glacial Drift Across New Jersey, and the Adjacent Parts of New York and Pennsylvania," read at the Wilkesbarre meeting, in May, 1877, and published in the *Transactions*, vol. vi. 467, was one of the earlier notices of the great terminal moraine in the eastern United States; and its importance in the history of geological science is now recognized by all glacialists.

The most valuable and notable results of the geological survey under the direction of Dr. Cook for so many years, may be thus summarized: The differentiation of the three green-sand marl-beds and their relation to the betterment of the soils; the geological structure of the overlying formations, and the guide to abundant supplies of excellent water in the southern part of the State; the description and accurate survey of the clay-beds, and their value as sources of refractory materials; the explanation of the structure of the magnetic iron-ore beds, and the methods to be followed in locating mines; the drainage of the Pequest and of the Passaic River valleys; the surveys of the water-sheds of the northern part of the State,

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pointing to the great natural reservoirs of water for city supplies; and the topographic maps, unsurpassed in their accuracy and detail of delineation, of the surface features of the whole State.

The distribution of useful information which would enable the miner to follow his vein or ore-bed, or discover new locations, and the farmer to better his condition through larger harvests, and whatever would make any of his fellow citizens more capable of doing their work well was the great end, not only of his geological studies, but of all his many engagements and activities.

Intensely loyal to New Jersey, he may be said to have been its foremost citizen; yet he was not limited by political boundaries in his aspirations for scientific knowledge, or in his interest in those who could be served by him. He was so busy with the practical development of what lay in the line of official duty that he had little time for the more purely theoretical or speculative in geological science.

He was an active member of the American Association for the Advancement of Science, and a frequent attendant at its meetings. He was president of the section of geography and geology at the Cleveland meeting in 1887. He was a member of the American Philosophical Society, and of the Academy of Natural Sciences in Philadelphia. In 1887 he was elected a member of the National Academy of Science. He took an active part in the International Geological Congress, and was the reporter on the Mesozoic for the American Committee of the Congress. His work, both in geology and in scientific agriculture, was widely known; and its substantial and valuable results gained for him a national reputation as one of the leaders in science. He was esteemed highly for the character of his work, as well as for his sterling integrity and unselfish devotion to scientific pursuits.

The degree of doctor of philosophy was given him in 1856, by the University of New York; that of doctor of laws by Union College in 1865.

His last illness was of short duration, and he was confined to his house for a day only. Heart-failure was the cause of his death, which occurred Sunday, September 22, 1889. His widow, two sons and two daughters survive him. In the death of Dr. Cook this Institute loses an honored member, a State its most useful citizen, and science an eminent leader. He has left for us the noble heritage of a well-rounded, Christian character for our consolation and incentive to duty.

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NOTES ON THE ADDITIONAL DIAPHRAGM IN THE HOWELL ROASTING FURNACE.

BY CHARLES W. GOODALE, BUTTE CITY, MON.

(Colorado Meeting, June, 1889.)

IN the course of the preparation of my paper on "The Occurrence and Treatment of the Argentiferous Manganese Ores of Tombstone District, Arizona" (*Trans.*, xvii., 767), my attention was called to two patents which have been issued for the use of a diaphragm at the upper end of a Howell furnace, whose importance merits more than a passing reference.

Mr. J. K. Pardee invented and first put into use in the Algonquin mill, at Philipsburg, Mon., in 1880, a "new and useful improvement in rotary ore-roasting furnaces," which was intended "to increase the capacity, effectiveness and working economies of such furnaces." The following description and accompanying figures are taken from the patent papers, dated December 21st, 1880:

Fig. 1 is a side elevation of the feed-end of the Howell rotary furnace, with parts broken away, the better to exhibit my invention. Fig. 2 is an end elevation of the same. Fig. 3 is an end elevation of the same, showing a modification of my invention.

In operating a furnace of this kind, of the capacity of one ton of ore an hour, it was found that for the best work the two fire-places (one at each end of the revolving cylinder) consumed about ten cords of wood every twenty-four hours, and that about twenty-four tons of silver-ore were introduced through the feed-pipe. It was also found that of the amount of ore, fully one-third did not go through the revolving cylinder, but went into the dust-chambers about the feed-end of the furnace or cylinder, and this ore going into the dust-chambers was found not to be sufficiently chloridized for amalgamation, but had to be re-roasted. I then caused to be made, out of boiler-plate iron, a segment of a circle, the height of which was about one-third of the diameter of the cylinder or furnace, the circumference of the circle being smaller than the circumference of the furnace, so that about a half-inch space would be left between the plate and the furnace, when said plate or segment was in position. This segment or plate of iron was then fastened to the bottom of the feed-pipe, as shown in the drawings, wherein A represents the revolving cylinder or furnace; B, the fire-place and dust-chambers at the feed-end of said furnace; C, the inclined feed-pipe, and D, the diaphragm, partition, or plate fixed within said furnace, A. With the diaphragm or plate, D, in the position shown, the furnace was run for about a month, and it was found that with the furnace in best working condition only four cords of wood every twenty-four hours were consumed, that no fire was required at the feed-end of the furnace, that not one-twelfth of the ore

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passed into the dust-chambers, and that the chloridization of the ore was more complete than ever before. The advantages of this device are, that when the

Fig. 1.

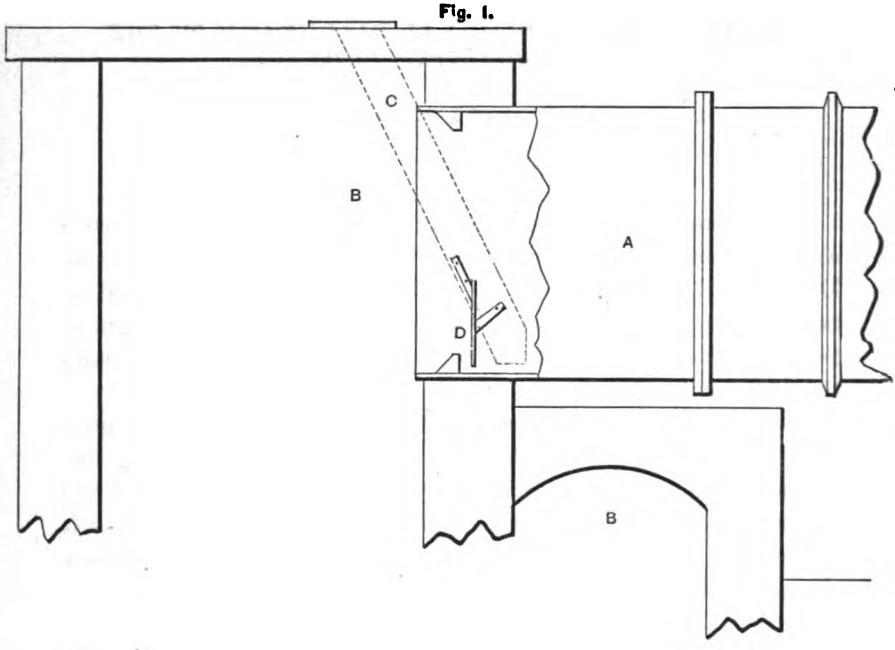


Fig. 2.

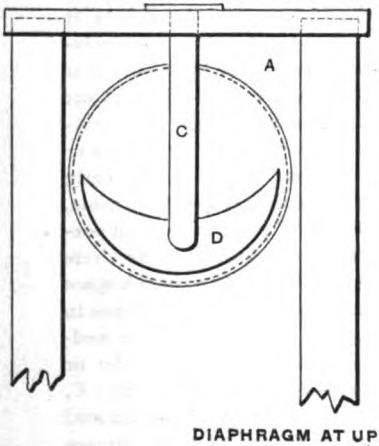
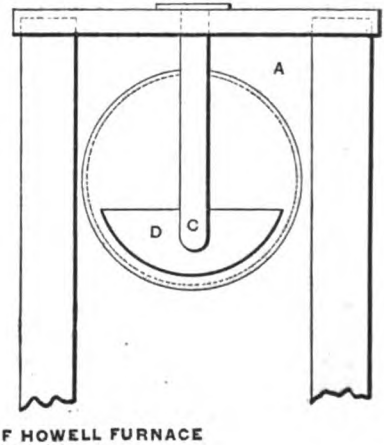


Fig. 3.



DIAPHRAGM AT UPPER END OF HOWELL FURNACE (PARDEE PATENT).

diaphragm or plate, D, is placed in position with the furnace, A, the cold crushed ore introduced through the feed-pipe, C, drops into an eddy in the cylinder, and is

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not for some time exposed to the flame and draft, said flame and draft being directed towards the upper part of the furnace, A, by the said diaphragm, D. The said crushed ore, therefore, has time to become heated, and its particles to become aggregated before it is carried up high enough by the rotary movement of the furnace to drop through the moving current of flame and air, so that the danger of being carried by the draft into the dust-chambers is avoided. By the use of this diaphragm, D, the auxiliary fire for roasting the air-floated crushed ore is also dispensed with, as the heat and products of the combustion from the main fire-box are retained longer and made more effective in the furnace. Consequently, the expense of building and attending to an auxiliary fire-chamber is avoided. This diaphragm, D, also prevents, as has been set forth, the making of an excess of flue-dust, not more than one-tenth or one-twelfth of the amount of ore passing through the furnace, thus preventing what has hitherto been a great expense and loss in the working of ore-roasting furnaces. This diaphragm, D, may be constructed of wrought- or cast-iron, and may be made in several forms, as a segment of a circle, as shown in Fig. 3; as a lune, as shown in Fig. 2, or in other convenient form. It can be fastened to the feed-pipe, C, as shown in Fig. 1, or to a suitable iron frame, and be suspended from above, and be braced and held in the desired position by suitable braces, and can be arranged to be raised or lowered by suitable devices.

I claim as new and desire to secure by letters patent, the combination with the rotary furnace, A, and feed-pipe, C, of the diaphragm or partition, D, substantially as herein shown and described

A Howell furnace, with the Pardee patent diaphragm, has been used for several years by the Moulton Mining Company at Butte, and has given good results.

Mr. L. M. Rumsey, President of the Granite Mountain Mining Company, is the inventor of a different form of diaphragm, which has been in very effective service in the mills of the company at Granite, Mon. From his patent papers, bearing date June 15th, 1886, the following description is taken :

Fig. 4 is a side elevation of a well-known form of furnace with my improvements attached thereto, parts being broken away more fully to illustrate my invention. Fig. 5 is a view in detail of my improvements.

In the well-known Oxland furnace an annular piece has been secured on the outside upper end of the cylinder and provided with a short tubular piece or pipe secured to the annular piece and projecting outwardly, or away from the cylinder. A crescent-shaped plate, or a plate of the form of a segment of a circle, has also been secured to the feed-pipe at the upper end of the cylinder on the lower side. In both of these instances the fine dust of the ore is not chloridized before it reaches the draft-line in the rotary cylinder, and much of it is carried out through the dust chambers unroasted, and is lost. To obviate these difficulties, as well as to dispense with the fire at the upper end of the cylinder, is the object of my invention.

Referring to the drawings, in which like letters of reference indicate corresponding parts, my improvement is shown as applied to the well-known Brueckner or Howell furnace. A is the fire-chamber; B, the rotary cylinder, and C, the mechanism for feeding the pulverized ore. D is a tube or pipe with a funnel-shaped mouth for conveying the ore into the upper end of the cylinder. The cylinder is

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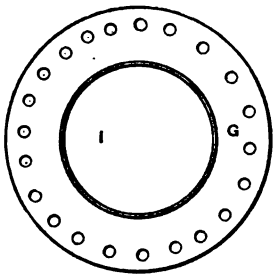


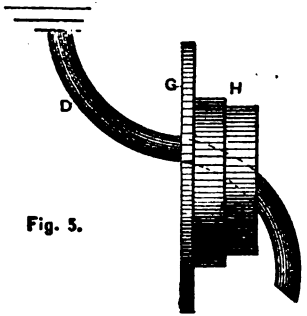
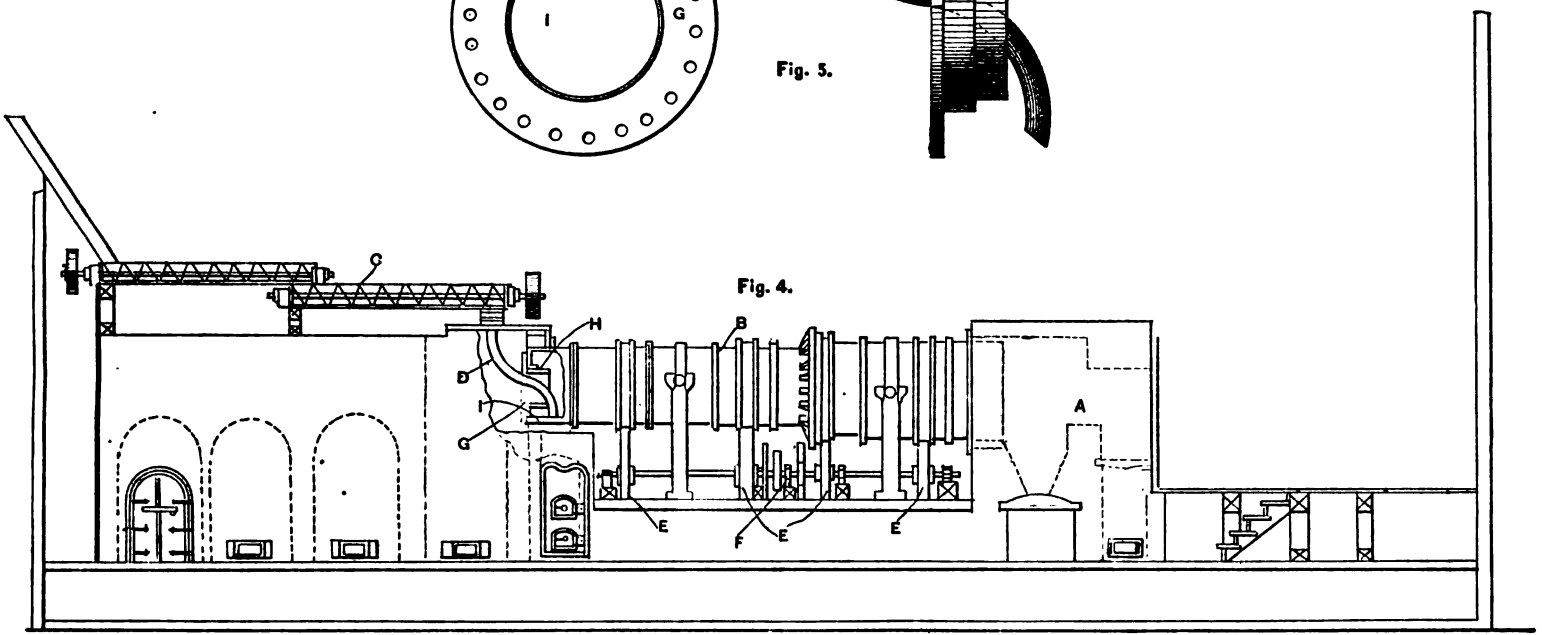


Fig. 5.

Fig. 4.



DIAPHRAGM AT UPPER END OF HOWELL FURNACE (RUMSEY PATENT).

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rotated by friction-wheels, E, and the driving shaft and gearing, F, or in any other suitable or known manner. The structure just referred to is well known and need not be further described in detail. To the upper end of the cylinder I secure a flanged annular piece, G, which is constructed of metal of the proper thickness, and is bolted, riveted, or otherwise secured, to the end of the cylinder. This ring-like piece, G, has an inwardly-projecting annular flange or tubular portion, H, thus forming on the inside upper end of the cylinder an annular chamber, I, which, being out of the draft-line of the cylinder, receives and holds the fine dust of the ore until it is roasted or chloridized, when, as before stated, it falls by gravity to the bottom, and is discharged with the heavier portions of the ore into the pulp-bin or receiver.

It will be seen that the feed-pipe is placed diagonally across the opening in the annular piece, G, and that it delivers the pulverized ore or pulp at the bottom of the cylinder. I prefer to construct this feed-pipe as shown in the drawings—viz.: somewhat curved, and with its lower end partly cut away, or with a section removed therefrom, as at A, on the side next the chamber, I. The object of this construction is to deliver the pulverized or stamped ore, or the bulk of it at least, into the chamber, I, instead of into the body of the cylinder, so that it will not fall to the lower end and pass out until it is thoroughly roasted.

The feed-pipe may be round, square, or of any desired form. I prefer to have the inwardly-projecting flange portion, H, about twelve inches in length in a sixty-inch cylinder, but I do not restrict myself to these proportions. The flange is preferably so located that it forms a chamber about one half of the area of the cylinder.

It will be seen by my improvement, that I am able to deliver into the cylinder and hold the lighter and finer particles of the dust of the ore out of the draft-line thereof until they are roasted or chloridized, and it will be found that all dust going into the flue chambers will be chloridized as high as, or higher than, that passing through the cylinder.

My invention also does away wholly with the use of an auxiliary fire at the small end of the furnace, and also decreases largely the amount of fuel used in the principal furnace at the lower end of the cylinder. Having thus fully described my improvement, what I claim as new, and desire to secure by letters patent, is:

1. A rotary furnace for roasting or chloridizing ores, provided at its upper or receiving end with closed internal annular chamber, to receive and retain the dust-particles of the ore until they are chloridized, substantially as described.

2. A rotary furnace for roasting or chloridizing ore, provided at the upper end of the rotary cylinder with the annular part, G, having inwardly-projecting annular flange, H, forming the closed internal dust-chamber, I, substantially as set forth.

3. A rotary furnace for roasting or chloridizing ores, provided with a feed-pipe extending through the upper end of the cylinder, said pipe having a portion of its lower end, on its side toward the end of the cylinder, partly open or cut away, substantially as set forth.

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THE WEAR OF RAILS AS RELATED TO THEIR SECTION.

BY P. H. DUDLEY, NEW YORK CITY.

(Ottawa Meeting, October, 1889.)

THE present paper was suggested by the paper read by Mr. R. W. Hunt at the New York Meeting, in February last, on rail-sections (*Trans.*, xvii., p. 778), in the discussion of which I brought forward some of the facts and observations here stated. Professional duties having prevented me from writing out my part of that discussion in time for its publication in the volume containing Mr. Hunt's paper, I take the opportunity to present it in separate form and greater fulness.

The striking features of the series of rail-sections submitted by Mr. Hunt are their shallow heads and heavy bases. In these respects they are quite similar to the section of the 80-lb. rail, which I designed in 1883 to meet the requirements of the traffic of the New York Central and Hudson River Railroad. Before the adoption of the 80-lb. rail, this road had modified its 65-lb. section from a shallow head with rounding sides and quite heavy base to one having a deep head and thin base. All the leading railroads had changed from the earlier sections of steel rails to those having deep heads and thin bases, conforming to the recommendations of the Committee on Rails of the American Society of Civil Engineers.

It is but proper to say that the committee's investigations upon the wear of rails were made upon the rails manufactured under a practice quite different from that which obtained after the committee's recommendations were carried into effect. The results in actual service have been contrary to what was expected.

Fig. 1 shows my 80-lb. section of 1883.

Before designing this rail I had taken, at different times, a series of diagrams of the condition of tracks, with my car, on the New York Central and Hudson River Railroad, all of the New England railroads, and all but one of the trunk lines. My diagrams gave me an opportunity which could not otherwise have been obtained to see the wear of great numbers of sections of rails under widely varying conditions of traffic.

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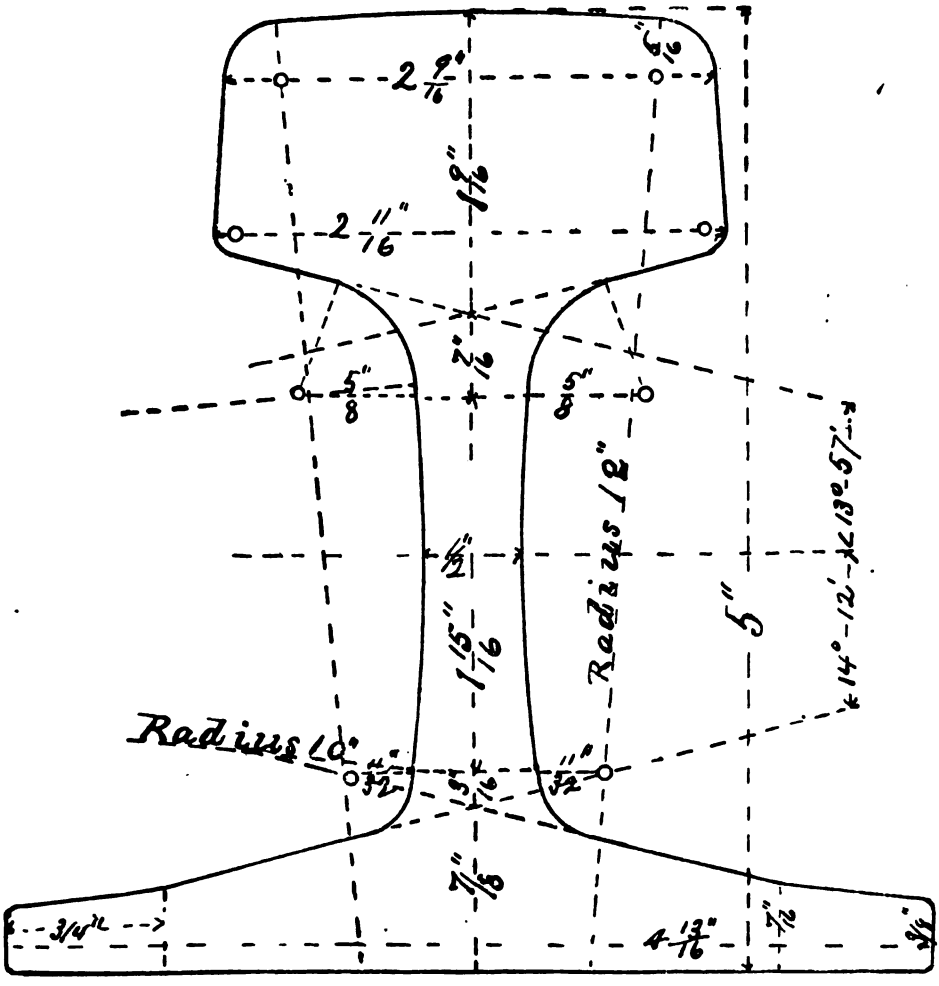
In nearly all cases it was observed that the deep-headed rails wore more rapidly than the shallower-headed; and, furthermore, did not wear as smooth, the metal on the surface of the rail flaking out in

FIG 1.



Section of 80-lb. Rail. 1883.

thin irregular patches to the depth of $\frac{1}{100}$ of an inch, or more, instead of wearing off smooth, as in the older rails.

The crystallization usually appears much coarser in the deep than in the shallow heads, and a thicker layer of metal intervenes between the crystalline groups or aggregates. This is illustrated in Fig. 2,

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which represents a longitudinal section of a tire of recent steel, polished and prepared, and magnified about 30 diameters.

Fracture usually occurs between the crystalline groups, not through such a group. In rolling a tire or rail, a group of these small crystals is elongated and flattened in a plane parallel with the surface. The crystals are first thinned, then loosened, by wear; and finally they flake out, leaving a very uneven surface on the tread of the tire or the head of the rail. The depth of these depressions is, as I

FIG. 2.

have said, about $\frac{1}{100}$ of an inch. They are found to range in number from 6 to 15 per square inch. Figs. 3 and 4 are reproduced from photographs of two steel tires, representing right and left car-wheels, which have thus flaked out.

Now the early steel rails did not flake out, but wore smooth. It is evident that the chemical composition of two rails may be nearly identical, and yet the difference in the grain of crystallization may give them different physical properties.

It was the consideration of these features which led me to design the broad and shallow head of the 80-lb. rail shown in Fig. 1. My first object was to lessen the pressures per unit of contact, thereby reducing the rate of wear of the rail-head and the tires.

The rails of which I had diagrams weighed 56, 58, 60, 65 and 67 lbs. per yard and ranged in height from $3\frac{3}{4}$ to $4\frac{1}{2}$ inches. In the track of the Boston and Albany Railroad there were several miles of 72-lb. rails, $4\frac{1}{2}$ inches high.

The diagrams of the condition of track showed conclusively that none of the rails were stiff enough to carry the traffic for any length of time and keep their surface without constant attention.

The rails examined in the tracks, with the aid of the track deflection apparatus of my car, to test the condition of their surface, were

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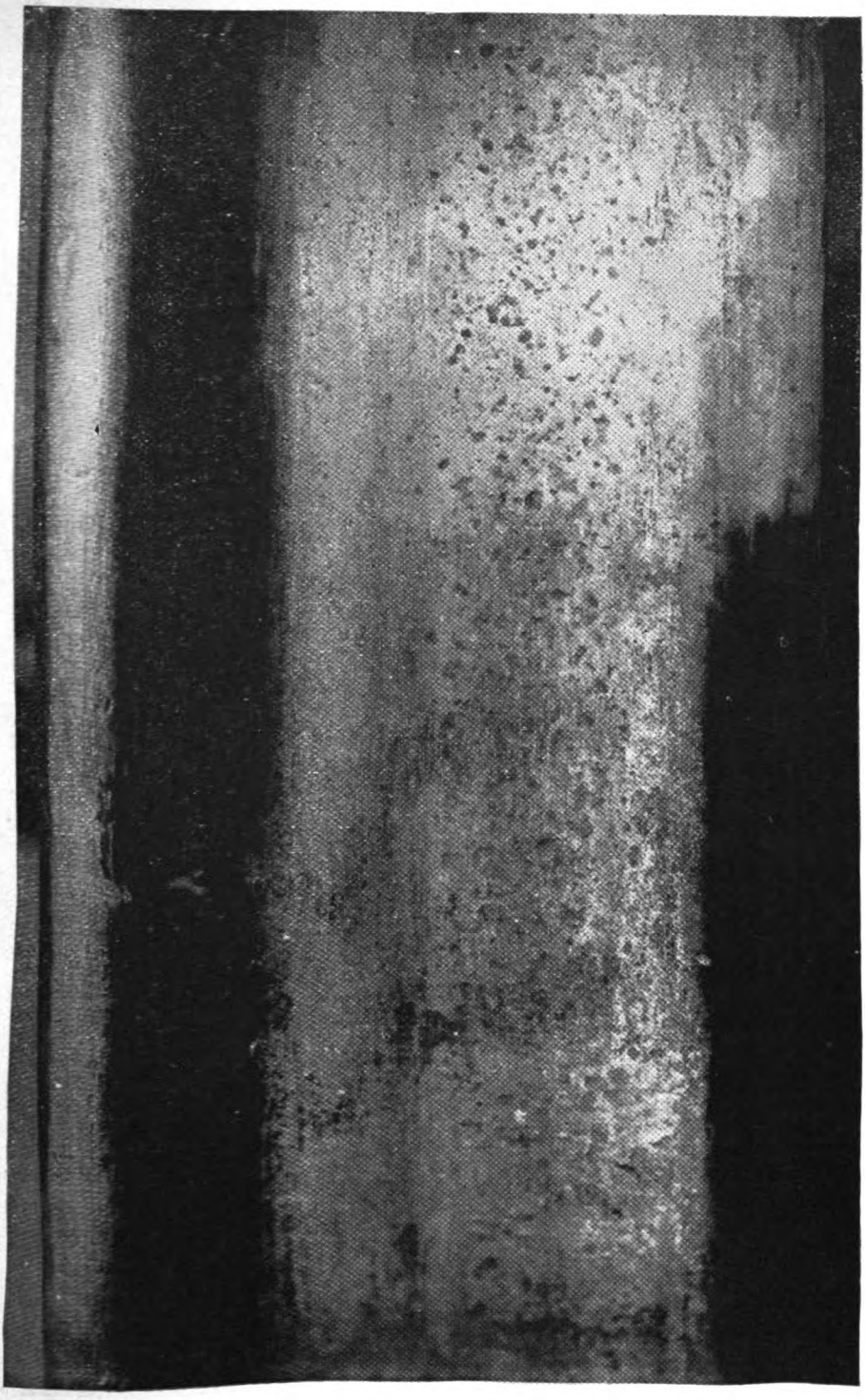

FIG. 3.—TIRE OF RIGHT-HAND WHEEL, No. 211.

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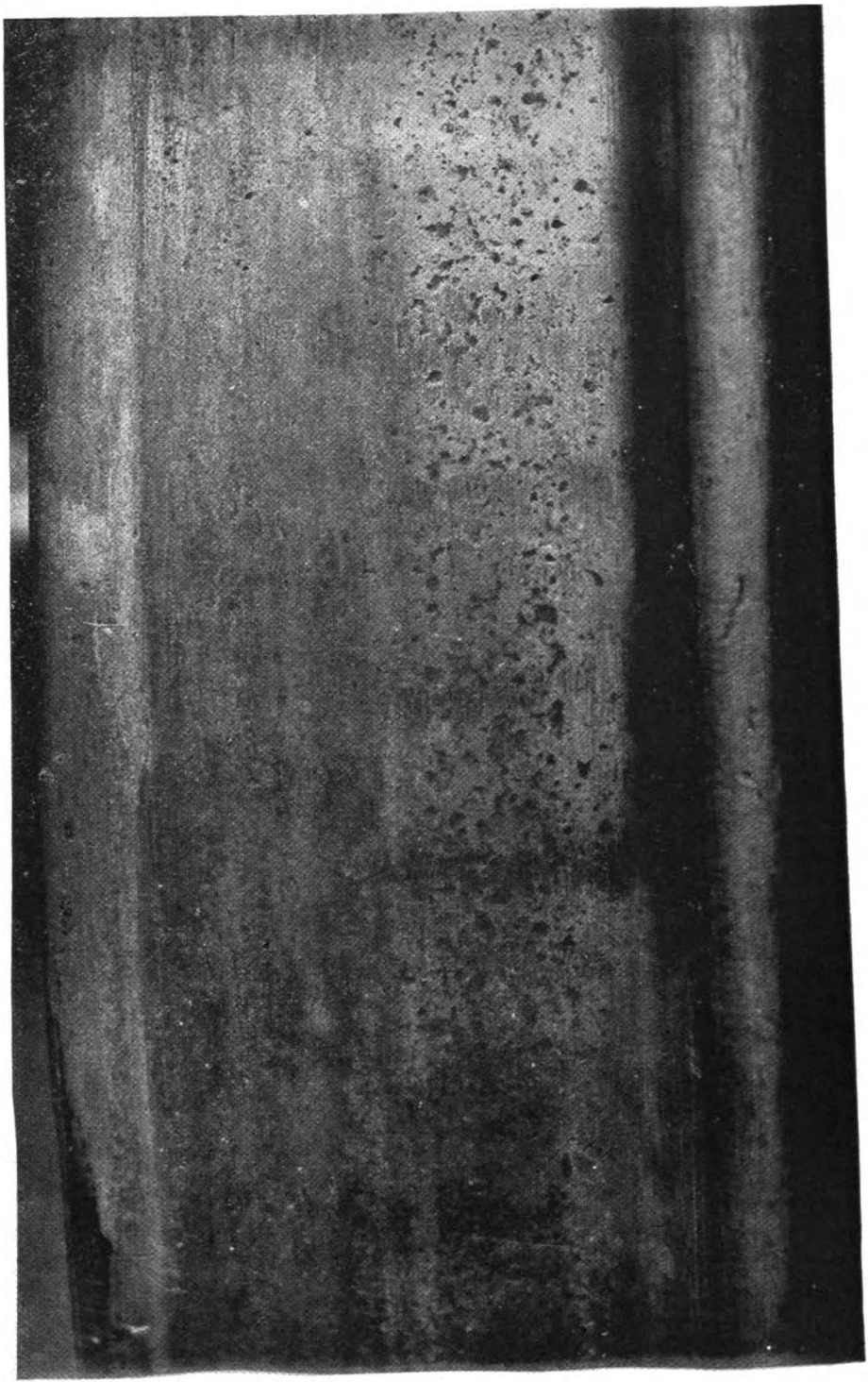

FIG. 4.—TIRE OF LEFT-HAND WHEEL, No. 212.

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found to have definite forms of permanent set, which could be directly traced to want of care in the track, or, in some cases, want of care in manufacture.

Rails which were low at the joints and high in the center, being the most frequent, I called the first form of permanent set; rails low

FIG. 5.

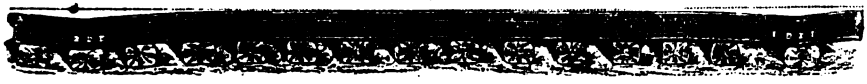


FIG. 6.

FIG. 7.

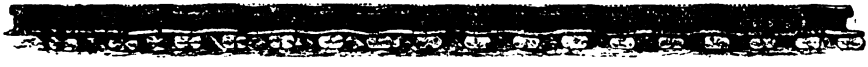


FIG. 8.

FIG. 9.

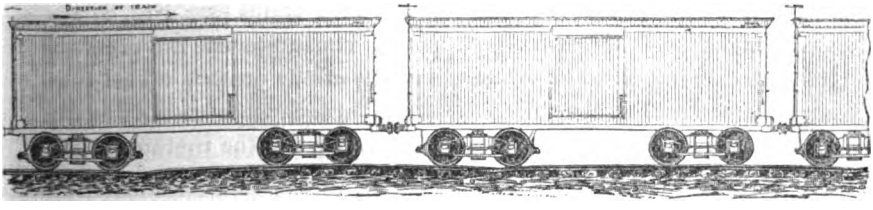


FIG. 10.

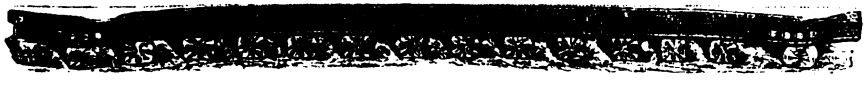
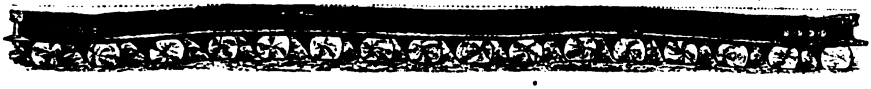


FIG. 11.



at the joints and center but high in the quarters, I called the second form; rails, the surface of which was a series of short waves, I called the third form.

Figs. 5, 6 and 7 illustrate these three forms.

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Combinations of the first and third, and the second and third forms were frequent. Fig. 8 shows the long waves which are found to take place in the best tracks under the weight of the cars, from want of stiffness of the rails and equal tamping of the ties. Fig. 9 shows the destructive effect of car-wheels upon rails of the first form of permanent set. One-half the length of the rail being an up-grade and the other half a down-grade, the receiving-end of the up-grade part is hammered down by the descending wheels. Figs. 10 and 11 show the effect upon the first and second forms respectively of trains running in the direction shown in Fig. 9. Trackmen are unable to surface such injured rails unless they are removed from the track, the ends sawed off and redrilled.

As a rule, rails laid with opposite joints show predominantly the first form of permanent set. In rails laid with alternate joints, on the other hand, the second form predominates, provided the section of the rail, joint-fastening, and labor expended in maintenance are sufficient to maintain the track under the traffic. If any one of the three requisites be lacking, rails of the second form sooner or later pass to the first form of permanent set. This occurs very soon in planked or paved stations and at road crossings. If the joints deflect $\frac{3}{16}$ of an inch or more under passing trains, the receiving ends of the rails will be cut out as shown in Fig. 10.

The deflections of rails under the wheels of trains (see Fig. 8) are more extensive than is generally supposed. The subject has not received the attention due to its importance, in view of our present wheel-tonnage. In low and light sections the weight of each wheel is not distributed over five or six ties, but is concentrated, for the instant, upon two or three—two, if the wheel is between them; three, if the wheel is over a tie. The second or third tie, as the case may be, is relieved of pressure, the base of the rail being under compression instead of tension, as would be the case with a rail of proper stiffness. The concentration of the wheel-weight over a small area of ballast quickly disturbs it; the spikes are started, the ties become loose, and are rapidly abraded under the rails, while the labor to maintain the track is much greater than would be the case with stiffer rails. This has been well demonstrated by experience with the 80-pound rail. In its design the stiffness of the section was an important feature, for the reasons above stated. Up to the present date it is stiffer than any 85-pound section yet produced. The following table gives its deflections under loads:

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Tests of the 80-Pound Rail of the New York Central and Hudson River Railroad for Deflections Upon Supports 30 Inches Apart, by the U. S. Testing Machine, at the Watertown Arsenal, June 21 and 22, 1888.

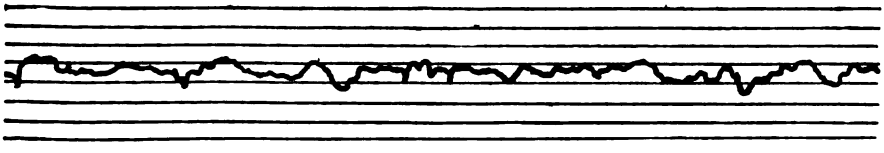
Load applied in Pounds.	Deflections in Inches.		Load applied in Pounds.	Deflections in Inches.	
2,000	0.		26,000	0.0258	
4,000	0.0022		28,000	0.0277	
6,000	0.0048		30,000	0.0300	set 0.0028
8,000	0.0071		40,000	0.0400	" 0.0036
10,000	0.0091	set 0.0019	50,000	0.0498	" 0.0048
12,000	0.0112		60,000	0.0597	" 0.0063
14,000	0.0133		70,000	0.0722	" 0.0085
16,000	0.0154		80,000	0.0855	" 0.0116
18,000	0.0175		90,000	0.1326	" 0.0447
20,000	0.0196		100,000	0.2488	" 0.1477
22,000	0.0218		110,000	0.4125	
24,000	0.0239		120,000	0.6440	" 0.5143

The tests from 80,000 to 120,000 were made June 22d, the rails resting 15 hours in the meantime. The rail only contained 0.32 of carbon, too soft for the best wear.

Two difficulties will be experienced in substituting the heavier, stiffer and broader-headed rails for the lighter sections in general use:

1. The want of experience on the part of the mills in finishing

FIG. 12.



heavy and stiff rails as perfectly as the lighter sections they are accustomed to rolling.

2. The wearing or shearing of the edges of the broader rails by wheels having hollow treads, worn by narrower-headed rails.

Both of these difficulties will disappear to a great extent upon a more general use of the broader-headed rails. As to the first, the finish of rails of the same section is not uniform among mills. As a rule too much camber is given to the rails when hot; and in taking out one long bend by the gag, two or more short ones are put in. In heavy rails these short waves are more noticeable than on light sections, and give an unpleasant tremor to the cars. The trackmen cannot remedy them. This is illustrated by diagrams of rails, taken by my car in the track, and represented in Figs. 12, 13 and 14.

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Fig. 12 shows the limited undulations of the best tracks, in 1887, the deflections rarely exceeding $\frac{1}{8}$ inch. The rails were well finished at the mill, and the joints do not show in the diagram.

Fig. 13 is the diagram from rails of the same weight and form as Fig. 12, and cared for by the same trackmen, but coming from another maker, and having a wavy surface and poor finish. On these the cars ride unpleasantly.

Fig. 14 is from new rails, not well straightened. They show short bends near the joints, and cannot be improved by trackmen. The cars ride as badly as on worn-out rails.

The difference in the finish of the rails is so distinct as to enable me often, where the rails are new, to tell from my diagrams the name

FIG. 13.

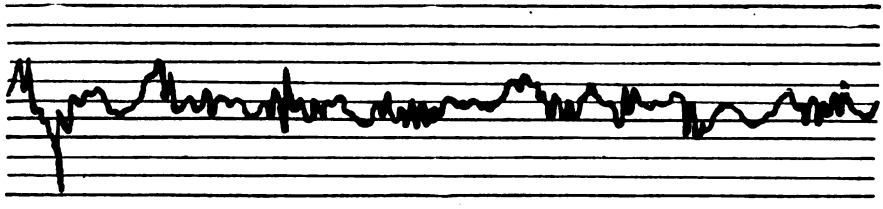
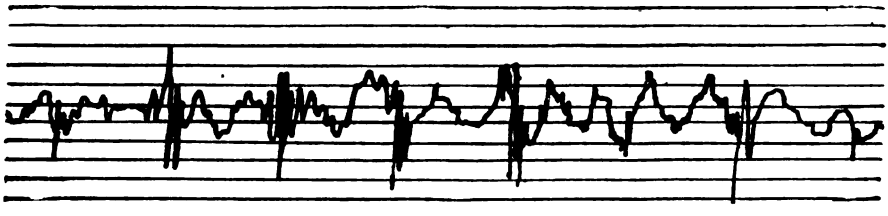


FIG. 14.



of the maker. To the credit of the mills, I should say that there has been a decided improvement in the finish of rails during the past few years.

As before observed, too much camber is given to the heavy rails when hot. Some of it is left in them when they cool. It should be the other way. The head of the rail might well have a true concave curve of an inch versed-sine, which would readily "surface," and tend to check for a long time a permanent set in an opposite direction.

But little experience has yet been accumulated as to the effect of the 80-pound rails on the service of tires. The records of the tires of locomotives Nos. 80 and 503 of the New York Central and Hudson River Road are as follows:

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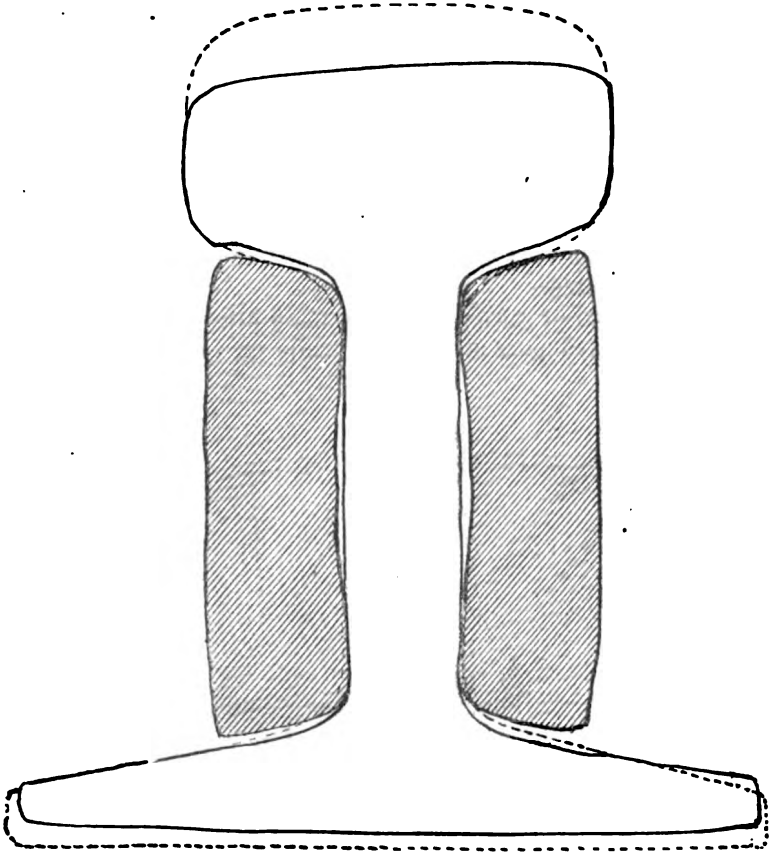
No. 80 has run 42,908 miles, making 5760 regular starts and stops. Figs. 17 and 18 are sections showing the wear of these tires.

No. 503 has run 54,390 miles, making 16,280 regular starts and stops.

The proportional wear of the tires is less than the average for the class of service performed on narrower-headed rails. The radius of

FIG. 15.



the curve of the worn tread of the tires so far is 13 inches, one inch more than that of the head of the rail.

The abrasion of ties under the 80 pound rails ($4\frac{13}{16}$ inch base) is very much less than the rate under the 65-pound rails ($4\frac{1}{2}$ inch base), the decrease being principally due to the increased stiffness of the rails. The latter feature has also checked to an unexpected degree the rate of oxidation of the 80-pound rails in the tunnels, as

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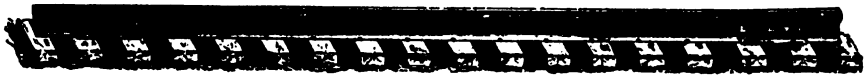
compared with that of the 65-pound rails. The rails are so stiff that after four years' service, the deflections of the rail have not detached the first scale of oxidation, which has served, in a measure, as a protection against further oxidation. Where the rails rest on the ties, abrasion has kept them bright, and here some loss of metal

FIG. 16.

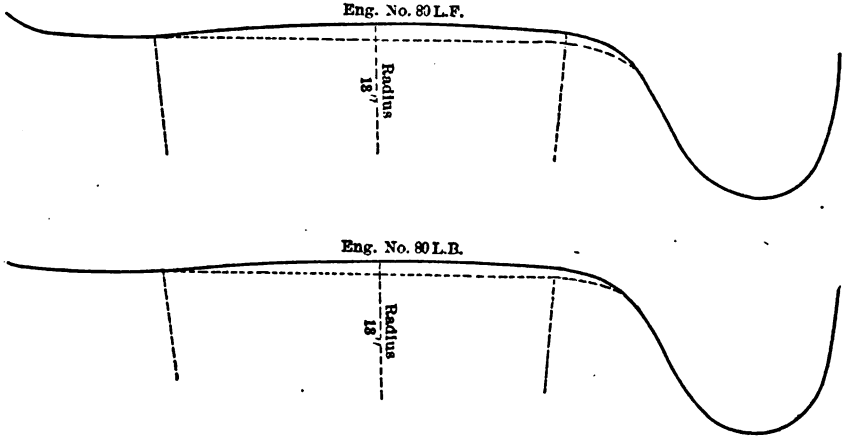


occurs—about $\frac{1}{40}$ of an inch in four years, which is only about $\frac{1}{4}$ the rate on the 65-pound rails, although the annual tonnage passing over the rails has largely increased.

The fillet of $\frac{5}{8}$-inch radius on the under side of the head of the 80-pound rail was deemed necessary to prevent the head from canting over in worn sections.

A large number of worn sections, having a fillet of $\frac{5}{16}$-inch radius, were found canted over on tangents, while on curves it was very

FIG. 17.



Sections of Tires of Engine No. 80, showing Wear.

common to find the heads canted out of their normal position. A $\frac{1}{4}$-inch fillet gives less support than the $\frac{5}{16}$-inch.

Fig. 15 is the section of a 65-pound rail after five years' service. It shows the wear, under heavy traffic, on the edges of the fish- or angle-plates, and of the base of the rail on the ties, of rails having

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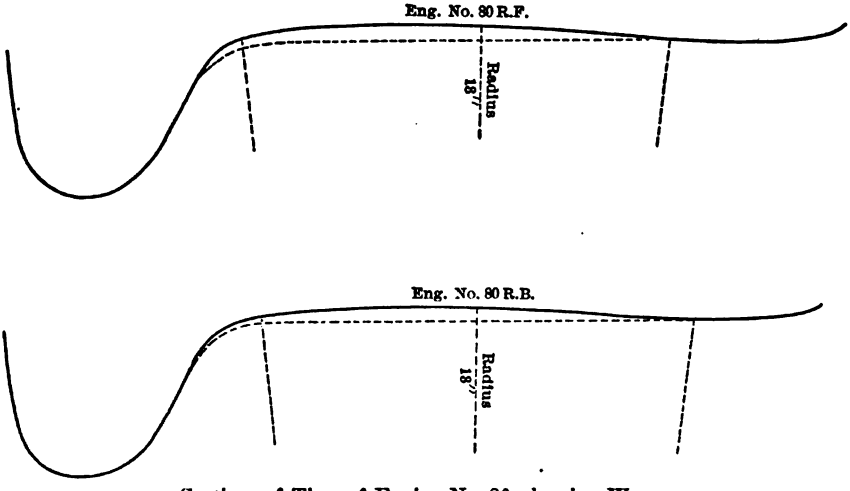
permanent set and large deflection. It is only the central portions of the fish- or angle-plates which wear on the edges. The ends, being unworn, appear firm to the trackmen. The bolts may be screwed up solid; yet the ends of the rails are free to go down and up under passing trains and are more or less cut out like those shown in Figs. 9, 10 and 11. The dotted line under the base in Fig. 15 shows the loss of metal from wear on the joint-tie. On a suspended joint, the loss of metal is not at the end of the rail but over the tie.

FIG. 18.



Sections of Tires of Engine No. 80, showing Wear.

Fig. 16 represents the best condition of track shown, in this instance, with suspended joint. It may be taken to represent also the best condition of rails with supported joints. The rails are in perfect surface to the eye, and liable only to the waves shown in Fig. 8, which should be prevented by vigilant care in surfacing and track-maintenance.

POSTSCRIPT.

Soon after the distribution of the preliminary pamphlet edition of the above paper, I received the appended letter and tables from Mr. Fred. A. Delano, Engineer of the Bureau of Rail and Rail-Joint Inspection, Tests and Records of the Chicago, Burlington and Quincy R.R. Co. The tables furnish such important data concerning the stiffness of rails as modified by the quality of the metal, that I have

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TABLE I.

Transverse Tests of 85lb Rails for the C. B. & Q. R.R., made with the United States Testing Machine, at the Watertown Arsenal, Mass., September 20, 1889. Supports 30 inches apart.

	DEFLECTION IN DECIMALS OF AN INCH.										PERMANENT SET IN DECIMALS OF AN INCH.									
	McClure section.					Delano section.					McClure section.					Delano section.				
Watertown No....	54	58	60	56	Aver'ge	55	59	57	53	Aver'ge	54	58	60	56	Aver.	55	59	57	53	Aver.
Heat No.....	5567	5602	5642	5621		5117	5071	5068	5115											
Load in lbs.																				
2,000	0	0	0	0	.....	0	0	0	0	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
4,000	.0022	.....	.....	.....	.....	.....	.....	.....	.0032	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
6,000	.0047	.....	.....	.....	.....	.....	.....	.....	.0058	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
8,000	.0069	.....	.....	.....	.....	.....	.....	.....	.0078	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
10,000	.0089	.0093	.0093	.0091	.0091½	.0093	.0094	.0094	.0099	.0095	.0002	.0004	.0003	.0003	.0003	.0003	.0004	.0002	.0013	.0005½
12,000	.0110	.....	.....	.....	.....	.....	.....	.....	.0120	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
14,000	.0130	.....	.....	.....	.....	.....	.....	.....	.0141	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
16,000	.0149	.....	.....	.....	.....	.....	.....	.....	.0161	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
18,000	.0167	.....	.....	.....	.....	.....	.....	.....	.0181	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
20,000	.0186	.0193	0.190	.0190	.0189¾	.0190	.0193	.0197	.0200	0195	.0004	.....	.....	.....	.....	.0008	.....	.....	.0014	.....
22,000	.0208	.....	.....	.....	.....	.....	.....	.....	.0220	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
24,000	.0228	.....	.....	.....	.....	.....	.....	.....	.0240	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
26,000	.0241	.....	.....	.....	.....	.....	.....	.....	.0259	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
28,000	.0260	.....	.....	.....	.....	.....	.....	.....	.0278	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
30,000	.0278	.0289	.0282	.0283	.0283	.0283	.0288	.0290	.0297	.0289½	.0004	.....	.....	.....	.....	.0005	.....	.....	.0016	.....
32,000	.0294	.....	.....	.....	.....	.....	.....	.....	.0313	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
34,000	.0310	.....	.....	.....	.....	.....	.....	.....	.0330	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
36,000	.0329	.....	.....	.....	.....	.....	.....	.....	.0348	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
38,000	.0347	.....	.....	.....	.....	.....	.....	.....	.0365	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
40,000	.0366	.0379	.0373	.0370	.0372	.0371	.0381	.0384	.0383	.0379¾	.0006	.0008	.0005	.0007	.0006½	.0006	.0007	.0006	.0019	.0009½
42,000	.0383	.....	.....	.....	.....	.....	.....	.....	.0400	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
44,000	.0400	.....	.....	.....	.....	.....	.....	.....	.0419	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
46,000	.0417	.....	.....	.....	.....	.....	.....	.....	.0438	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
48,000	.0436	.....	.....	.....	.....	.....	.....	.....	.0455	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
50,000	.0454	.0470	.0464	.0462	.0462½	.0460	.0472	.0474	.0473	.0469¾	.0007	.....	.....	.....	.....	.0008	.....	.....	.0020	.....

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52,000	.0471								.0491												
54,000	.0490								.0509												
56,000	.0508								.0524												
58,000	.0525								.0543												
60,000	.0543	.0560	.0549	.0554	.0551½	.0550	.0563	.0568	.0562	.0560¾	.0010						.0008			.0022	
62,000	.0560								.0582												
64,000	.0577								.0600												
66,000	.0595								.0618												
68,000	.0612								.0636												
70,000	.0630	.0650	.0642	.0660	.0645½	.0636	.0655	.0663	.0655	.0652¼	.0013						.0009			.0030	
72,000	.0651								.0676												
74,000	.0670								.0695												
76,000	.0690								.0717												
78,000	.0708								.0737												
80,000	.0726	.0735	.0737	.0767	.0741¼	.0725	.0751	.0760	.0758	.0748½	.0021	.0011	.0019	.0052	.0025¾	.0012	.0014	.0011	.0044	.0020¼	
82,000	.0747			.0793		.0745			.0782												
84,000	.0764			.0817		.0765			.0804												
86,000	.0782			.0843		.0782			.0829												
88,000	.0802			.0869		.0800			.0852												
90,000	.0822	.0833	.0833	.0900	.0847	.0821	.0850	.0870	.0880	.0855¼	.0030	.0026	.0031	.0099	.0046½	.0020	.0022	.0037	.0076	.0038¾	
92,000	.0847	.0851	.0858	.0931		.0840	.0873	.0894	.0905												
94,000	.0865	.0871	.0880	.0958		.0860	.0892	.0918	.0929												
96,000	.0876†	.0891	.0900	.0988		.0880	.0912	.0944	.0953		.0030										
98,000	.0896	.0912	.0921	.1020		.0900	.0940	.0972	.0981												
100,000	.0916	.0933	.0946	.1054†	.0962¼	.0922	.0960	.0995	.1024	.0975¼	.0037	.0038	.0051	.0162	.0072		.0035	.0040	.0067	.0121	.0065¾
102,000	.0939	.0957	.0976	.1097		.0944	.0983	.1025	.1048												
104,000	.0961	.0979	.0998	.1130		.0964	.1008	.1051†	.1080												
106,000	.0982	.1002	.1020	.1178*		.0987	.1034	.1085	.1105												
108,000	.1010	.1024	.1048	.1223		.1009	.1060	.1120	.1134†												
110,000	.1039	.1050	.1077	.1285	.1112¾	.1033	.1089	.1168	.1180	.1117½	.0068	.0062	.0089	.0298	.0129¼		.0052	.0074	.0140	.0199	.0116¼
112,000	.1068	.1077	.1109	.1355		.1060	.1123	.1213	.1227*												
114,000	.1096*†	.1100†	.1138	.1430		.1083	.1150†	.1268*													
116,000	.1140	.1127*	.1165†	.1578		.1109	.1184	.1340	.1306												
118,000	.1181	.1157	.1198	.1726		.1133*†	.1225*	.1433	.1349												
120,000	.1221	.1188	.1240*	.1950	.1399¾	.1170	.1275	.1547	.1410	.1350½	.0161	.0108	.0158	.0850	.0319¼		.0099	.0164	.0412	.0326	.0250¼
122,000	.1300	.1223	.1284			.1202	.1370	.1680	.1500												
124,000	.1363	.1259	.1322			.1230	.1460	.1821	.1562												
126,000	.1453	.1323	.1370			.1260	.1607	.1992	.1665												
128,000	.1580	.1428	.1430			.1302	.1743	.2126	.1824												
130,000	.1743	.1540	.1524			.1358	.1925	.2338	.1973		.0578	.0360	.0343				.0190	.0693	.1077	.0782	
132,000		.1695	.1640			.1433															
134,000		.1942	.1740			.1502															
136,000		.2050	.1850			.1620															
138,000		.2125	.2040			.1750															
140,000		.2360	.2197			.1895							.0899				.0618				
"	after 5 min.	.2478																			
"	" 10 "	.2520										.1205									

NOTE.—* Indicates point at which scale starts off from vicinity of middle bearing. † Indicates point of approximate elastic limit.

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TABLE II.

Prepared by Mr. Fred. A. Delano, in part from Table I.

Showing the comparative deflections and set at 100,000 pounds; approximate elastic limits; test for hardness; height and weight of the different rails designated in Table I. The added carbon and phosphorus determinations were made by the manufacturers of the rails.

		Heat No.	Watertown No.	Deflection at 100,000 pounds.	Set at 100,000 pounds.	Approximate elastic limit.	Test for hardness with pyramidal cutter.	Actual height in inches.	Actual weight per yard.	Carbon.	Phosphorus.	
McClure....		5567	54	.0916	.0037	114,000	21.90 "A"	5.17	84.60	.39	.092	
		5602	58	.0933	.0038	114,000	21.29	5.18	83.50	.42	.098	
		5642	60	.0946	.0051	116,000	26.83	5.18	82.08	.48	.100	
		5621	56	.1054	.0162	100,000	21.64	5.18	84.75	.41	.092	
Average.....		.....	.....	.0962	.0072	111,000	22.91½	5.18	83.73	.42¼	.095½	
Delano..	"B"	5117	55	.0922	.0035	118,000	26.99	5.02	85.75	.50	.108	"B"
		5071	59	.0960	.0040	114,000	21.40	5.00	81.00	.46	.102	
		5068	57	.0995	.0067	104,000	22.02	4.99	81.72	.47	.106	
		5115	53	.1024	.0121	108,000	25.73	5.02	85.50	.52	.093	
Average.....		.....	.....	.0975	.0065¾	111,000	24.03½	5.01	83.49	.48¾	.102¼	

"A." The higher figures denotes the harder rail.

"B." First heat rolled.

NOTE.—The "Delano" has greater "deflection" and less "set" than the "McClure" at 100,000 lbs. load.

obtained Mr. Delano's permission to publish them in connection with my paper.

It is proper to say here that my statement with regard to the stiffness of the New York Central 80-pound rail was made at the New York meeting of the Institute in February, 1889, prior to the rolling of the Chicago, Burlington and Quincy experimental sections. The tests of those sections, as given in Tables I. and II. (pp. 238, 239, and 240) indicate an excellent quality of metal, having a high elastic limit, probably exceeding 65,000 pounds per square inch, for the metal in the base of the rails. Comparing these tests with the single one of the low-carbon 80-pound rail of the New York Central, it will be seen that the metal in the latter takes a permanent set at a much lower strain.

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Tests of the same grade of metal in rails nearly as heavy indicate that the elastic limit of the metal in the base of the rail will not exceed 40,000 pounds per square inch.

The average of the results of the Delano section shows greater stiffness than the single test of the New York Central, except at 10,000 pounds. The "moment of inertia" of the latter is 26.96.

At 20,000	pounds	the	Delano	section	is	nearly	1	per	cent. stiffer.
" 30,000	"	"	"	"	"	4	"	"	
" 40,000	"	"	"	"		6	"	"	
" 50,000	"	"	"	"		6	"	"	
" 60,000	"	"	"	"		6.6	"	"	
" 70,000	"	"	"	"		10	"	"	
" 80,000	"	"	"	"		14	"	"	
" 90,000	"	"	"	"		55	"	"	

The elastic limit of the steel was passed between loads of 60,000 and 70,000 pounds.

The 80-pound section, the tests of which were reported in my paper, was a piece from the first 1000 tons rolled. The percentage of carbon specified was from 0.45 to 0.50. Why the makers allowed it to go below, I have never understood. The carbon has since been specified as 0.60 to 0.65, the stiffness and hardness of the rails being increased accordingly.

Under the present heavy wheel-tonnage, rails are wearing more rapidly than formerly; the subject is being studied more carefully, and, as a consequence, rails of better physical properties are demanded.

One railroad company, making tests of rails of its standard section as rolled by three different mills, was surprised to find a variation of over 20 per cent. in stiffness among the rails of the different makers. In hardness, the variation was even greater.

It was clear, in this case, that the difference in physical properties could not be ascribed to differences in section, but was due to difference in quality of metal. This has since been verified by comparisons of the wear of the rails in the track under practically the same tonnage.

The following is the letter accompanying the foregoing tables, received from Mr. Delano:

CHICAGO, December 5, 1889.

MR. P. H. DUDLEY, C.E.

Dear Sir: I have read with a good deal of interest the paper on rails which you read at the Ottawa meeting of the American Institute of Mining Engineers; and in view of the fact that your paper is having pretty wide circulation in the railroad jour-

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nals, I venture to correct a statement which you make in regard to the stiffness of your New York Central 80-pound rail. Last March the Chicago, Burlington and Quincy had 1000 tons of each of two different sections of 85-pound rail rolled at South Chicago for experiment; and, after the order was completed, four samples, four feet long, were cut at random from rails on which the heat-number was plainly stamped (*i.e.*, four of each section), and sent to the Watertown Arsenal for test. A partial report of the test, made in September, has been received, and I have tabulated it (see Table I.). This, and the accompanying smaller sheet (Table II.), will, I think, satisfy you that each of these 85-pound rails is stiffer than the New York Central 80-pound section, although you will note, from the actual weights of the samples tested, that the average weight of each of the groups is appreciably less than 85 pounds per yard. The vertical moments of inertia for these two sections are respectively 28.61 for the "McClure" and 27.34 for the "Delano" section. How do these compare with your New York Central 80-pound section?

I think you will be interested in seeing that the "McClure" section, although having a higher moment of inertia, does not, from the Watertown tests, prove to be an appreciably stiffer rail. This is what I worked for and what I expected. In regard to the chemical analysis given, I warn you not to build too much upon it, as it is simply the determination of the rolling-mill from the test-piece of the same heat-number as the rails tested. The Watertown people have promised to make analyses from the actual samples, and these should be of value.

Yours truly,
FRED. A. DELANO.

CONCENTRATION BEFORE AMALGAMATION FOR LOW-GRADE, PARTIALLY-DECOMPOSED SILVER-ORES, WITH NOTES ON THE GEOLOGY OF THE FLINT CREEK MINING DISTRICT.

BY C. W. GOODALE, BUTTE, MONTANA, AND WILLIAM A. AKERS, PHILLIPSBURG, MONTANA.

(Colorado Meeting, June, 1889.)

THE treatment of silver-ores of too low a grade to justify either smelting or preliminary roasting, and yet not "free" enough to permit of raw amalgamation, has ever been a serious problem. As bearing upon it, the operations of the Combination Mining and Milling Company of Black Pine, Deer Lodge county, Montana, during the past year may not be without interest. It is not claimed that the process to be described is not susceptible of great improvement, nor is any claim of originality made for it; but its use is, as yet, not very general, and it seems to offer a means of profitably working the large class of partially-decomposed argentiferous ores which, by reason

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of their low grade, demand some cheap, continuous process for their profitable treatment.

As introductory to an account of the occurrence of the ores, a brief general statement regarding the geological peculiarities of the entire district may not be amiss, since it is the home of two of America's greatest silver producers, the Granite Mountain and the Bi-Metallic mines, and bids fair to achieve even more prominence in the mining world than it already possesses.

On the east side of the Flint Creek Valley, 4 miles back from the base of the foot-hills, and elevated about 2500 feet above them, is a rugged chain of mountains, heavily timbered in places, and covered with immense masses of granite which have broken off from greater elevations and rolled down their sides. Within a few hundred feet of the summit of one of these peaks is situated the Granite Mountain mine, and immediately below it, on the same vein, the Bi-Metallic. The strike of this vein is nearly due east and west, and its dip from 8° south to 8° north of the vertical. Near its apex the walls are of very hard granite, dark-gray in color, and bear evidence of movement in the polishing of their surfaces. In width the vein at this point varies from 2 feet to 18 feet, the ore being separated from both walls by a clay selvage, and the chutes dipping westerly. In depth the vein widens out from 50 feet to an extreme of 150 feet, the pay being confined to a ribbon of ore running through it of from 2 feet to 22 feet in width. A number of times in the past it has been thought that the vein forked, but in every instance these supposed splits proved, upon investigation, to be merely due to immense "horses" of granite. At no point does the vein actually crop, nor is it common to find the pay-chutes nearer than 100 feet to the surface. On the western slope of the mountain, where the vein is not water-shedded, the ore gives unmistakable evidences of leaching, and, in consequence, the first chute encountered in running the adits was too low in grade to pay; but after passing through this, and driving several hundred feet on a pinched and perfectly barren vein, a second and very rich chute was encountered. Upon the discovery of this second or "Bonanza" chute, as it was called, miners were at once put stoping, and from 1435 tons treated in a custom-mill the company realized $273,897.81, and the erection of a 20-stamp dry-crushing mill was at once begun. In grading out for the retort-house the top of another chute was encountered, and this, in depth, has proven to be one of the richest and most extensive yet discovered in the mine. Subsequently, the Bi-Metallic Mining Company, con-

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trolled by the men who had been instrumental in the purchase and development of the Granite Mountain mine, located a shaft distant over 1000 feet from the extreme western workings of the latter, and sent it down to the 200-foot level before cross-cutting. The vein was there, and drifting soon caught the pay-ore; and now, with a 3-compartment shaft sunk to the 800-foot level, and long drifts in both directions from the shaft every 200 feet, it bids fair to equal, if not excel, the Granite Mountain mine itself. Connection has been made between the drifts of these two companies, and the joint levels open up a trifle over 6000 feet on the vein.

About 2 miles west of the Granite Mountain mine, and on a general northerly and southerly course, is a contact between two distinct geological formations. On the west the formation consists of limestone, shale and quartzite of the older series, probably Cambrian, while east of this contact, including the higher portion of the mountain range, the country-rock is an eruptive granite of a much later geological age than the bedded formations along its flanks. This is made manifest by the crushing and crumpling of the bedded rocks and the penetration of them by spurs of granite at various points immediately west of the contact. Further proof of the eruptive character of the granite is furnished by the remarkable chemical changes produced in the limestone and shales along and near this contact, and extending, in some instances, 1000 feet or more west of it. Along this metamorphic belt, the purer bands of limestone have been changed to marble, and the impure or shaly lime-beds made vitreous in some places and at others changed to epidotic and garnetiferous beds. The granite partially overlies the bedded rocks along its west flank. This is to be observed at many places where explorations have been made along the contact. The dip of the latter is invariably to the east under the granite. Gash-veins extend westerly from this contact as far as the chemical and thermal influences of the granite have made any marked changes in the crystalline structure of the beds, but beyond these limits of metamorphic changes the fissures disappear. There is a broad zone of mineralized limestone extending along the contact from 20 feet to 60 feet in width, containing large bodies of manganese-ore carrying more or less silver.

The Hope mine, in the limestone west of the contact, is the oldest in the district. A section of the hill, as revealed by shafts and diamond drill-holes, shows the first 50 feet to be a yellow, magnesian limestone, under which, and immediately over the vein, is a layer of shale from an inch to a foot in thickness. The width of the vein is

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very variable, running from a mere seam to 24 feet. Forming the footwall is a stratum of mottled limestone from 2 feet to 3 feet in thickness, and under this a stratum of very hard, white limestone. Next follows a 7-foot zone of black limestone; then from 10 feet to 25 feet of mottled limestone; then from 2 feet to 3 feet of barren quartz, and underlying all a very compact limestone, of a jet black color, which the deepest diamond drill-hole, 642 feet from the surface, failed to pierce. The strata dip to the north at an angle of 32° from the horizon. No fossils or evidences of organic life have ever been found in the limestone. In the old stopes, opened fifteen or twenty years ago, small stalactites and stalagmites have formed. The strike of the vein is east and west, and in dip it is conformable to the stratification of the country. Developments in this mine, which have been quite extensive, have revealed four principal faults, all at right angles to the strike of the vein, the latter term being employed merely as a matter of convenience, and not to define its precise character. Going west on the strike of the vein from the extreme eastern workings, the ore drops 92 feet at the first fault, has its regular strike for a distance of 60 feet, faults up 65 feet, is unbroken for 50 feet, drops 18 feet, has its strike again for 45 feet, and then drops 265 feet. In all four of these faults the plane of fracture is more or less inclined, and in every instance the hanging of the fault drops.

All the cañons opening into the Flint Creek Valley bear testimony to the erosive effects of the glaciers which have ground their way down them, and small terminal moraines have been deposited at their mouths. In the Boulder cañon, the largest tributary to the Flint Creek, opening into it about 12 miles north of the town of Phillipsburg, the marginal moraines are very heavy, and the terminal moraine, located at the confluence of the two cañons, is 3 miles long and from 1 to 2 miles wide. This moraine is elevated but slightly above the surrounding country, and is composed of round granite-boulders, few of them being of any great size. The largest one measured contained 1000 cubic feet. There is nothing to lead one to suppose that the Flint Creek Valley was ever the bed or track of a glacier, nor is there any way of determining the character of the rock under the valley, which varies from 1 to 2 miles in width; but on the western foothills extensive ruins of red sandstone are to be noticed.

In the range of mountains west of the Flint Creek Valley, and distant from it about 10 miles are situated the mines of the Com-

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bination Mining and Milling Company. The formation in this locality varies considerably, being in some places a friable, laminated sandstone, and in others metamorphosed into a quartzite. In places it is slightly micaceous. In all cases the strata dip in a westerly direction, and at an angle of 20° from the horizon. The strike of the vein is north and south, and in its dip it is conformable to the stratification of the sandstone.

Going west on the dip of the vein from its apex, five faults have been encountered thus far, the strike of which is with the strike of the vein, or north and south. At the first one, 100 feet down on the vein from its apex, the ore is faulted up 35 feet, and for 100 feet it has its regular dip; then it drops 15 feet, is unbroken for 40 feet, is carried up 10 feet, continues without a break for 50 feet, drops 10 feet, is regular again for 130 feet, and is then raised 25 feet. At right angles to this series are two other faults, 50 feet apart, the vein being dropped 20 feet at one and elevated the same distance at the other. Eighteen minor faults, where the vein is only shifted 2 or 3 feet, have been encountered during the past year. In all these displacements, great and small, the plane of fracture is always more or less inclined, and in every instance the hanging-wall of the fault has gone down. Whenever our workings on the vein have encountered a fault, there being no evidence of any horizontal squeezing, we have always followed this law, and have been invariably successful in finding the vein beyond it. In a number of instances upon striking a fault, we have continued our regular developments in opening up the mine without going to the expense of raising or sinking simply to find the vein beyond it, knowing that it was sure to be either above or below its former position as the inclination of the fracture indicated, and laying out our work accordingly. While this practice may be condemned as being too risky, in actual results it has been the means of saving the company hundreds of dollars for dead work—an important factor in the commercial success of mining a low-grade ore. With the smaller breaks the fracture is clean and abrupt, the walls of the faults being polished by mutual abrasion, and pressing one against the other; but in the more important ones the distance between walls is from 1 to 3 feet, the fissure being filled with talc, decomposed country-rock, small boulders of sandstone and black manganese. In the last fault mentioned as crossing the strike of the vein this is not the case, the walls being from 6 to 10 feet apart, and the filling-matter consisting of a decomposed quartz gangue carrying from 25 to 75 ounces of silver and $\frac{1}{10}$th of an ounce of gold to the

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ton. This fissure as far as developed seems to be bounded above and below by sandstone walls; although in every particular save this it is a regular fault, the vein south of it entering at its bottom and passing out of it to the north at its top. It would seem that the silver has concentrated in this crevice between the upper vein north and the lower vein south, and that the ore-body has been formed by substitution in a fault the sides of which were originally close together. In this crevice the ore is attached to the walls on all sides, and small pockets of it fill irregular shaped cavities in the walls.

Taking an average of the vein exposed during the past year, its width will not exceed 2 feet, its greatest size being 4 feet and its smallest 6 inches. It is usually laminated in structure, bands of from 6 to 18 inches of decomposed ore assaying from 20 to 200 ounces of silver to the ton being attached to both foot and hanging walls, and the middle portion of the vein consisting of quartz, more or less barren. It has been noticed that where the country-rock was hard and tight, the vein was always pinched and impoverished, and no change for the better occurs without a concomitant change in the walls of the vein.

The following analyses, made by Messrs. Von Schulz and Low, of Denver, from a sample of the mill battery for the month of October, 1888, will show the character of the ore:

Qualitative tests gave silica, carbonic acid, sulphur, iron, copper, lead, antimony, arsenic, zinc, manganese, silver, gold, a little chlorine, a little mercury, doubtful traces of cobalt, and very small amounts of calcium, magnesium and soda.

The same sample by quantitative analysis gave:

	Per Cent.
Silica,	84.09
Sulphur,	0.19
Iron,	7.05
Copper,	1.31
Lead,	1.01
Zinc,	0.30
Manganese,	0.35
Antimony,	0.66
Silver,	0.08
Arsenic,	0.05

In gold the ore runs 0.0000857 of 1 per cent., or 50 cents to the ton. In connection with the analysis of the Combination ores we quote the following remarks of President Richard Pearce: "The Combination ore is of a very curious nature. My attention was

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directed to it some six years ago, and I then discovered hübnerite (tungstate of manganese), and a very rare mineral, pseudomalachite (phosphate of copper), and these finds were noticed in the *Transactions* of the Colorado Scientific Society. The presence of P_2O_5 in the ore is suggestive of the origin of the deposit; it must have come from the adjoining rocks. We know of no phosphides which correspond with the arsenides of copper. Arseniates of copper can always be accounted for by the oxidation and alteration of enargite into arseniates." Our attention having been called to the presence of hübnerite by Mr. Pearce, a determination was made by Messrs. Von Schulz and Low from a sample of concentrates, 20 tons into 1, which gave 0.33 per cent. tungsten.

The Milling.

The mill, as originally constructed by Messrs. Fraser and Chalmers for the Black Pine Mining Company, was an ordinary 10-stamp, wet-crushing mill, with room left between the battery and settling-tanks for concentrating machinery should it be found necessary to add it. As illustrating the impractibility of working these ores by raw-amalgamation alone a short summary of the operations of the Black Pine Mining Company may be given.

The mill was started on the 16th of July, 1887, and the run lasted until the 14th of September of the same year. During this period there were treated 1178 dry tons of ore, assaying 17.5 ounces, and containing 20,615 fine ounces of silver. Of this amount there was recovered in the form of bullion 9482.90 fine ounces, or 46 per cent. of the battery assay.

We need only note the points of difference between the Combination Company's plant and the form of mill ordinarily employed for treating silver-ores of the class usually termed "free-milling." These, as is well known, are so-called more because their low grade precludes their profitable treatment by the various other more efficient but vastly more expensive methods, than because they are better adapted to that process.

The ore is stamped, passed over four Frue vanners, the light pulp that goes over their "tails" being settled in tanks, shovelled into pans, and there amalgamated, discharged into settlers, and the resultant amalgam strained, retorted and melted in the usual manner. An analysis of these concentrates for the month of October, 20 tons into 1, gave the following results:

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	Per Cent.
Silica,	82.50
Lead,	9.10
Copper,	8.22
Sulphur,	1.19
Zinc,	0.81
Silver,	0.54

Samples of the pulp going to the pans for the same month showed that it contained:

	Per Cent.
Copper,	1.05
Lead,	0.48
Zinc,	0.25
Sulphur,	0.09
Silver,	0.05

For the year ending May 31, 1889, the details of milling were as follows:

Dry tons crushed,		9,061.965
Average assay, ounces of silver per ton,		22.67
Gross contents of ore, ounces of silver,		205,434.75
Estimated per cent. of reduction,		80.70
Estimated product,		165,785.84
Dry tons of concentrates produced,		541.805
Average assay of concentrates, ounces of silver per ton,		136.17
Silver in concentrates,	73,777.55	
Silver in bullion,	97,660,60	
Total fine ounces,		171,438.15
Actual per cent. saved,		83.45
Total cost of milling,		$39,537.61
Average cost per ton,		$4.36

Batteries in service, 347 days, 5¾ hours.

Average ore crushed per stamp in 24 hours, 2.612 tons.

An analysis of the cost of treating one ton for the year gives the following figures:

Labor and superintendence,	$2.5821
Salt and other chemicals,	.2965
Fuel, at $1.00 per cord,	.1944
Castings and iron,	.3846
Oils and illumination,	.1066
Quicksilver,	.4915
Miscellaneous supplies and team in yard,	.3084
Total,	$4.3641

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When the mill was started by the present management, an assay of the escaping slime-water was made, showing the value of the slimes carried by it to be 52 ounces in silver to the ton, or more than double the silver contents of the original ore. A series of carefully conducted experiments were at once inaugurated, and from them it was learned that three tons of these slimes, dry, containing 156 ounces of silver, were passing through the waste-gate from the settling-tanks every 24 hours. These slimes were too light to permit of their being thoroughly settled by even the most extensive system of tanks, and to check this heavy loss, a China pump was placed in the last of the series of slime-tanks. This pump discharges into a small tank placed above and immediately in front of the mortars. It is only 1 foot deep, being made shallow in order to prevent the gradual settling of slimes in it, and the consequent necessity of cleaning it out from time to time. From the side of this tank, and near its bottom, is the 2-inch pipe for supplying the water to the stamps. The only purpose of this tank is to furnish water to the stamps at uniform pressure. By prohibiting the use of any clear water in the battery, the escape of these slimes was very materially reduced, but this of itself was not enough to stop all overflow, as it would be in an ordinary wet-crushing mill, for reasons which will be explained later on.

The immediate effect of turning back into the battery, say two tons every 24 hours of slimes carrying twice as much silver as did the original ore (and this proportion of values has always existed), was to heavily "salt" our battery-sample, and as the most rigid economy was essential to the commercial success of the property, the question of securing any accurate check on the mill became momentous. After careful consideration, it was decided to adopt the following system: Night and day samples are taken from the battery-launders in the usual manner, and are assayed daily with the other mill samples, but the results are corrected by the salting for the previous month. To determine this percentage, it is simply necessary to obtain the actual battery assay, and this is secured from the data furnished by the balance of the mill work, after the monthly averages are made up, in the following manner: The number of tons crushed, minus the number of tons of concentrates produced, must equal the number of tons amalgamated. The number of tons amalgamated, multiplied by the average assay pan-sample, must be equal to total silver contents of the pulp amalgamated. This, added to the total silver contents of the concentrates produced, secured in the same

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manner, gives the grand total of fine ounces of silver in the ore. This divided by the number of tons crushed gives the actual battery-assay. In this computation all tons are dry tons. The percentage of salting thus determined varies slightly, but the fluctuation is due more to errors in sampling than to any change in the actual amount of the salting. Usually, this is about 5 per cent.

But there was still a heavy loss in the escape from the settling-tanks of water carrying rich slimes, owing to the introduction of water at the heads of the vanners. To stop this, a small jet-pump, using steam from the boilers and supplied with 50 feet of steam-hose for a discharge, was placed at the tanks, and slime-water used in lieu of clear water for thinning down the pulp in the pans. While this corrected the evil, it also had the effect of increasing the difficulties in the way of concentration, as, in order to stop all overflow from the tanks, only as much water could be used on the heads of the vanners as was pumped into the pans, plus the evaporation.

After much experimenting it has been found, that with the limited quantity of water that can be used on the vanners, a speed of 180 revolutions of the crank-shaft per minute, with a belt travel of 3½ feet per minute, and an inclination of 3½ inches in the length of the machine, gives the most satisfactory results on these ores, although intelligent and constant adjustment is rendered necessary by reason of the changes occurring from time to time in the character of the pulp treated.

By reference to the details of milling for the year ending May 31, 1889, it will be noticed that the mill overran its assays 2.49 per cent. or 5144.63 ounces. As the actual battery-assay is determined by the assays of average pan and concentrate samples, and as the pan-sample is of necessity taken before the rich slime-water is pumped into the pans for the purpose of thinning down the pulp, the mill is bound to overrun, presuming perfect accuracy of sampling and assaying, exactly the amount of that part of the silver contained in the slime-water used in the pans which is amalgamated. As the greatest care is exercised to secure the most accurate sampling possible, a large portion of the amount that the mill overran may safely be credited to the use of slime-water in the pans.

The pan-charges have been changed constantly, according to the character of the pulp treated, and the amalgamation tests have been many and varied. The charge now being used, which seems to be the best tested, is 50 pounds of salt, 2 pounds of sulphuric acid, and ½ pound of cyanide, with 100 pounds of quicksilver strained in after

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the pan has been running 4 hours. The temperature of the pulp is raised by steam to 180° Fahrenheit, and the charge is run at 65 revolutions per minute for 8 hours. Settlers are run 14 revolutions per minute, and give good agitation with 3-inch shoes.

In the battery, 40-mesh brass-wire screen-cloth has been used during the past year, with the exception of about two weeks, when 30-mesh was tried by way of an experiment. After a thorough test it was found that the loss occasioned by higher tailings exceeded the profit accruing from increased crushing capacity, and the 40-mesh screens were replaced. The falling off in the savings during the time the 30-mesh screens were in use was largely owing to the fact, that the difference between the very finest of the slimes and the coarse particles that would pass through a 30-mesh screen was so great, that with the increased quantity of pulp to be treated by the vanners, they could not be made to do close and clean work.

The resulting concentrates have been shipped to a smelter for final treatment; but experiments are now in progress looking to their treatment at home, and it is highly probable that the necessary machinery will be added to the plant in the near future.

THE DISTRIBUTION OF PHOSPHORUS IN THE HUDSON RIVER CARBONATES.

BY INGERSOLL OLMSTED, MORRISTOWN, N. J.

(Colorado Meeting, June, 1889.)

MR. D. H. BROWNE'S paper upon "The Distribution of Phosphorus in the Ludington Mine" (*Trans.*, xvii., 616) has interested me very much, and perhaps I cannot give a better proof of that interest than an account of my work and observations in the same direction while chemist of the Hudson River Ore and Iron Company.

The ores of this company lie in a chain of low hills running northwest and southeast, about three miles back from the eastern bank of the Hudson river, below Hudson, N. Y. They occur in thick veins, with the characteristic Hudson river slate formation as hanging-rock and foot-wall, and dip about 45° east, outcropping in many places near the top of the western slopes.

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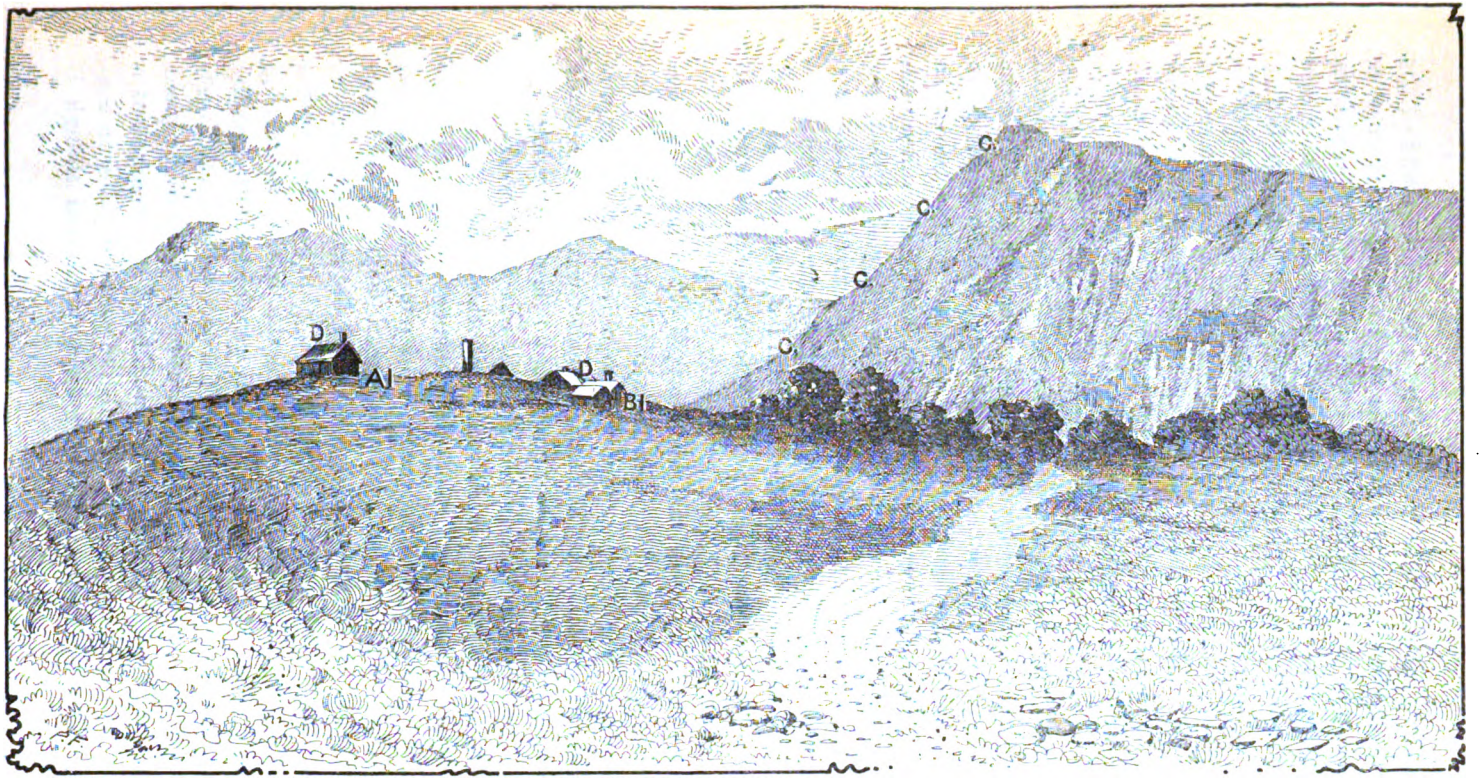

Sketch showing Location of the Mines of the Hudson River Ore and Iron Company. *A*, first opening, called Mine No. 2; Bessemer. *B*, second opening, called No. 3; Bessemer. *C*, *C*, third opening, called Mt. Tom; non-Bessemer. *D*, *D*, company's plant. The slope of Mt. Tom is made somewhat too steep in the sketch.

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The first opening, made in a low hill on the southern end of the property, was into a uniform Bessemer ore, carrying an average of 35 per cent. of carbonate of iron and 0.03 per cent. of phosphorus. The ore had a very fine, dense grain, and very dark lead color, and broke into flat slabs. It underlay the whole hill in a deep bed or pocket, and pinched out suddenly to the southward and eastward.

When this limitation had been developed by the drills, a second opening was made on the north end of the same hill, at B in the sketch, where the ground flattened out at the base of a rise of much greater elevation to the northwest.

As soon as the slope had been carried down under the hanging-wall, chambering was begun on each side of it, with the unexpected result of developing non-Bessemer ore upon the north side, while the south side remained Bessemer. The diamond drill-holes put down in advance of the northern workings showed the continuance of the non-Bessemer ores there; and the company stood face to face with the fact that the south end only of its property yielded Bessemer ore. The accompanying sketch shows the formation of the country and the openings. The vein on Mt. Tom lies upon the eastern side of the hill, as though it had originally lain flat, and, when the hill was upheaved, had been tilted up so as to form an obtuse angle with the adjoining surface.

When I took charge of the laboratory, No. 2 mine was almost exhausted; No. 3 and Mt. Tom had just been opened. I was informed that the Bessemer ore was so uniform as to require very little attention, but the non-Bessemer was so hopelessly irregular in chemical composition that no law governing the variations was discoverable. The deposit was understood to be double, to consist of two distinct ores, and the wonder was constantly expressed that no more of the Bessemer ore existed. The phosphorus was considered to occur as phosphoric acid, P_2O_5, and therefore not to be recognized by the eye, a condition of things which I could hardly understand, considering the irregularity.

To investigate the matter, I began by taking lumps of the roasted ore, crushing them roughly, and examining the mass under a strong magnifier. The impurities thus revealed were picked out atom by atom, and, when any particularly unfamiliar one was observed to persist, enough was collected for a phosphorus test. No familiar pink-and-white apatite grains were to be found, but I soon recognized a lead-colored mineral existing in thread-like veins or plates in considerable quantity; and a series of tests in this, mixed though it

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necessarily was with grains of ore, showed an average of over 3 per cent. of phosphorus. With this light I began an inspection of the daily shipments of non-Bessemer ore, armed with my magnifier, and the unrecognizable phosphorus-compound appeared everywhere. There I made, also, another observation. The ore presented several persistent appearances, such as:

a. { 1. A rough, coarse-grained, light red mass.
2. " " blue "

b. Slates and similar forms of ore, breaking into slabs.

These I have already described in a former paper.* The observation I made was, that the phosphorus was very abundant in *a*1; not quite so much so in *a*2, and scarcely noticeable in the B varieties. Analysis of samples of each of these varieties showed that the *a*1 was our richest ore, *a*2 the next, *b* the lean, and that the percentage of phosphorus followed the percentage of iron, the *b* varieties being lean Bessemer ore, existing in narrow, well-defined veins threading the ore-body. Here I was confronted with the fact, from my own analysis, that the drill-cores showed no constant relation between the percentages of iron and phosphorus; but this difficulty disappeared upon considering that, since the impurities (silica, carbonate of lime, lead, phosphorus, etc.) exist free, irregularly intermingled with the ore-grains, any small portion like six inches of drill-core cannot represent anything but that immediate piece of ore. The peculiar phosphorus compound in the soft, irregularly-grained ore was hard as flint, breaking out whole and fairly separate from the mass, sometimes in pieces as large as $\frac{1}{2}$-inch square by $\frac{1}{8}$-inch thick, but generally as small, thin plates or shot. In thread-like veins, it often ran side by side with veins of pure white silica or of galena. I considered it to be true apatite, though, being busy at the time and seeking the information only for my own pleasure, I only submitted its composition, aside from the phosphorus, to qualitative tests.

Following this substance into the Bessemer ore, I recognized it only in small quantity; in that from the No. 2 mine, which was slaty in formation, it was in the form of fine shot; in that from No. 3 mine, the *a*1 variety of ore began to mingle with the No. 2 mine slates, bringing with it its characteristic threads of phosphate, though

* "Notes on the Roasting of the Hudson River Carbonates," *Trans.*, xvii., 275.

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not frequently enough to affect much the percentage of phosphorus.

I next visited the workings to study the appearance of the natural non-Bessemer ore; but in a few days the openings on Mt. Tom were abandoned, and no further opportunity of observing the varying faces was offered. At my one visit, however, I found the face of the ore-body traced over with veins of the *b* varieties, varying from an inch to a foot in thickness. They were mainly near the floor, which was somewhat above the foot-wall, but whether they were always so or not, I had no certain testimony. Above and around them lay the rough, coarse, light-gray ore. At times, I was told, the slate, which is practically a lean Bessemer ore, broadens out into a body several feet in thickness, and is thrown out on account of its leanness. In a sample of this sent me for an iron-test, the iron existed as 24 per cent. carbonate, and 15 per cent. of some form insoluble in acids. I mention this last fact because some wiser geologist than I may thence draw some inference concerning the formation.

At this point circumstances directed my researches into another channel. The fact that this phosphorus compound was separable from the ore by the magnet, led to experiments in electric separation which are foreign to the present subject.

Without elaborating any exact theory of "isochemic lines," I have always considered that the south end of this property was not a separate ore-body, existing independently, but was the same as the north end, with less impurity; that the characteristics of the south end penetrated into the north somewhat in the form of lean Bessemer slates; that the fineness of the grain had some close connection with the presence of phosphorus, and that some such washing and irregular deposition as Mr. Browne suggests had taken place.

In the No. 2 mine ore and the Mt. Tom slate, scarcely any impurities are visible, the silica and the lime being in combination; but in the Mt. Tom ore of the "*a*" varieties, the mass is seamed and threaded in every direction with many minerals.

In the light of Mr. Browne's explanation, the "isochemic lines" to the contrary notwithstanding, I shall call this "irregular deposition by water," until the theory is overturned.

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CONCENTRATION OF LOW-GRADE ORES.

BY HENRY E. ARMITAGE, LAKE CITY, COLO.

(Colorado Meeting, June, 1889.)

THE object of this paper is to give a few useful hints on the concentration of low-grade ores. The machines that I shall speak of are, Cornish rolls, revolving screens, Hartz jigs, spitz-lutte, and the Linkenbach buddle (or Roberts's patent buddle, as it is called in Colorado). I think the reason why some of them have been discarded in favor of more recent inventions is that they have not been given a fair chance to do their work in a satisfactory manner.

Cornish rolls are usually looked upon as a machine designed simply to crush the ore as a preliminary to sending it to the sizing-screens, beyond which they have nothing to do with dressing the ore. On the contrary, the rolls give the key-note to the whole process. It is well known that one class of rock will crush differently from another, and from the result obtained by passing the ore through the rolls the size of the first revolving screen and the surface required on the jig-screens must be determined. If it is found that the mineral crushes clear from the quartz, a large mesh can be used on the first sizing-screen, and the tailings will not require recrushing; but if it is found that some of the tailings from the first jig are ragged (that is, are carrying off particles of mineral sticking on the quartz), it will be necessary to recrush them. Again, if a large percentage of the tailings carry off mineral, it will be more economical to commence with a finer mesh screen than to recrush the tailings.

Having determined the best mesh to use on the first sizing-screen, the different sizes on the other screens can be determined from it. I have found for practical purposes that, commencing with a 6-mesh, the next should be 8, then 12, and finally 18. The holes in the jig-screens should be a trifle larger than in the revolving screens. For instance, if the revolving screen is made of No. 14 wire, the corresponding jig-screens should be of No. 16 wire. The jig-screen surface required for each size made depends on the way

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the ore crushes in passing through the rolls, and this is shown by the quantity of crushed ore passing through each revolving screen. Supposing, for example, that the sizings are 6, 8, 12, and 18, and it is found that 40 per cent. of the crushed ore passes through the 6-mesh screen, 15 per cent. through the 8-mesh screen, 15 per cent. through the 12-mesh screen, and 30 per cent. through the 18-mesh screen; then, if three-compartment Hartz jigs are used, it will require two three-compartment jigs to treat the ore from the 6-mesh screen, one three-compartment jig to the 8, and one three-compartment jig to the 12-mesh. The 18-mesh jig catches everything that is fine enough to pass through the 18-mesh revolving screen, and it is necessary to separate the finer slimes from the ore before it is allowed to go on the jig. I have found that a spitz-lutte does this work most satisfactorily. As I have not seen one in any other mill in Colorado, I present a sketch of this very simple machine.

It consists of two inverted cones, one inside of the other. The inner cone is suspended from a frame by a threaded bolt fastened to a cross-brace, *aa'*, on the cone. The outer cone rests on the dead water tank, *cc'*. The ore and slimes after passing through the 18-mesh revolving screen, run into the inner cone and pass out through the perforations, *B*, at the bottom, into the space between the two cones. Here they encounter an up-current of water coming from the pipe *bb'*. This pipe is connected with a tank at sufficient height to give the required head. The finer slimes are separated and washed out at the point *cc'* where they run into the trough, *EE'*, surrounding the outer cone and are carried to the buddle-table by the spout, *F*. The heavier particles of mineral and quartz fall into the hopper, *A*, and are drawn off by the stop-cock, *G*, on to the jig. By raising or lowering the inner cone, by the screw and nut, *ee'*, the space at the point of discharge can be increased or decreased. It is essential that this space should be of the same area as the area of the space at the point *H*, so that the up-flow of water will be of uniform velocity from the starting point at *H* to the discharge at *cc'*. This flow is regulated by the valve, *K*, and by it the quantity of slimes sent to the buddle-table can be controlled. The stop-cock, *G*, is kept open only sufficiently to let the ore that falls into the hopper, *A*, run off to the jig.

This machine will take off all the slimes and let the ore of proper size go to the 18-mesh jig, the quantity being about 20 per cent. of the ore crushed. As this ore, being very fine, takes longer to settle and pass through the jig-screen than the coarser sizes, I find it

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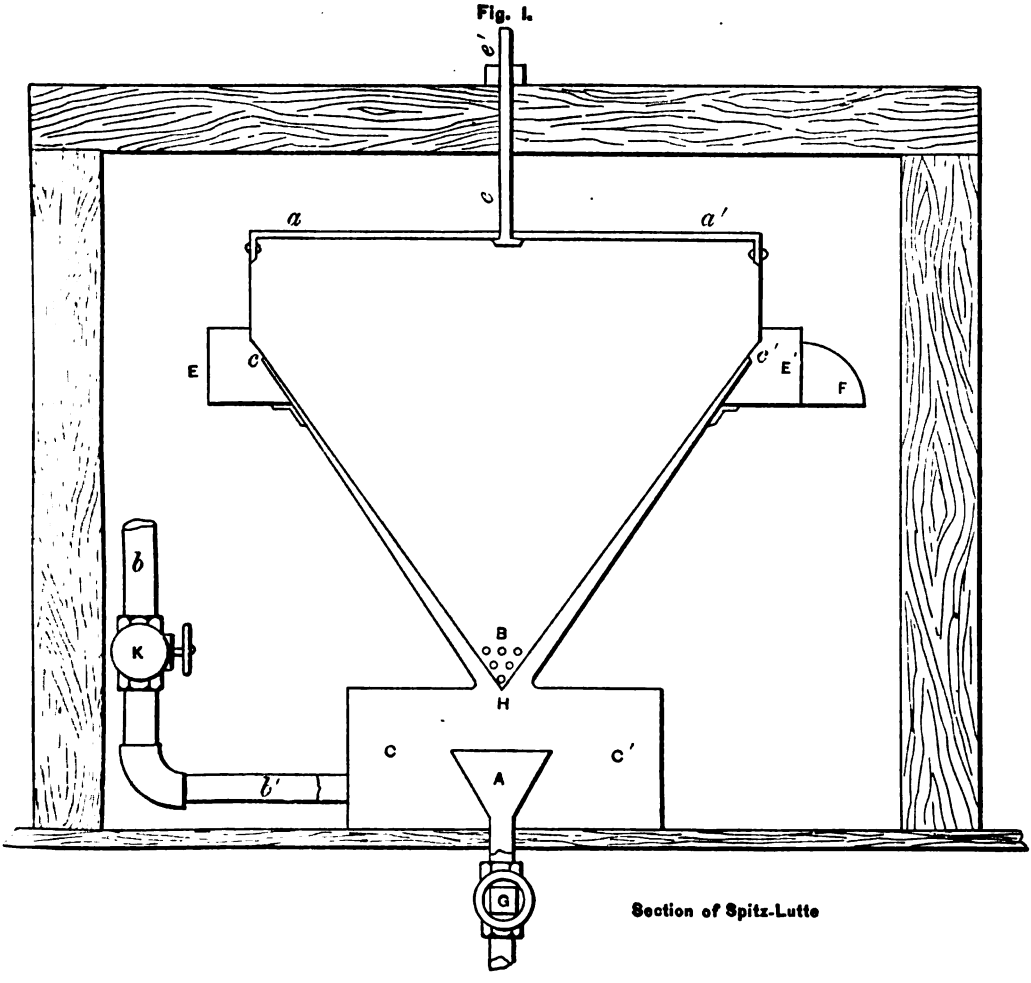
better to increase the number of compartments on this jig. This refers to ores that contain galena, gray copper, yellow copper, and iron pyrites, and from which it is necessary to save and separate these minerals. In doing this on a three-compartment jig, I found the

Fig. 1.



Section of Spitz-Lutte

copper minerals and iron pyrites were crowded out at the tail-gate before they had time to settle and pass through the jig-screen. But if the ore has to pass over five or even six compartments before reaching the tail-gate, it has had time to settle through the beds and the tailings will be clean.

The Linkenbach buddle was fully described in a paper by Mr.

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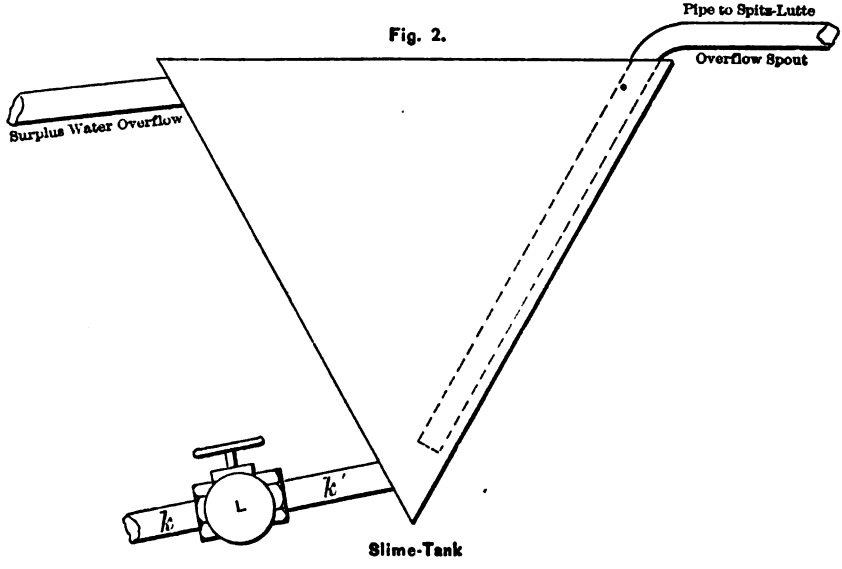
Richard P. Rothwell, read at the Boston meeting, February, 1883 (*Trans.*, xi., 475). It is a stationary table with rotating water sprays, which I consider an excellent apparatus for treating low-grade slimes, but, like the jigs, it must receive fair treatment to be able to do its work well. The slimes as they come from the spitz-lutte carry too much water. The whole of the water used on the revolving screens, together with that used in the spitz-lutte for washing out the slimes, would run on the buddle and prevent it from doing its work as it should. To prevent this excess of water from reaching the buddle, I let the slimes run into a hopper-shaped tank and draw them off from this to the buddle. By making the

Fig. 2.



slimes from the spitz-lutte discharge at the bottom of the tank, as shown in the accompanying sketch, very little can rise to the surface of the tank, and get away by the overflow, the slimes being sucked out by the pipe, *kk'*, as fast as they are delivered. The valve, *L*, must be left open sufficiently to allow the slimes to pass to the buddle without accumulating in the tank, the surplus water passing off at the overflow. The middlings from this table, consisting of the lighter ores, I carry to a Frue vanner.

The length of stroke given to the jig-plungers is a matter to be determined by experience. No rule can be given, since the difference of fit of the plungers in the jigs would change the effect of any given stroke.

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I do not think it is advisable to increase the number of strokes when the stroke is shortened; that is, to run the finer jigs at a higher speed than the coarser ones, as this has a tendency to disturb the bedding, though I believe it is done in many cases.

In bedding the jigs, the best plan is to bed each compartment with the same ore which that compartment is going to handle. In treating an ore containing galena, gray copper, and iron pyrites, bed the first compartment with galena, the second with a mixture of galena and gray and yellow copper, and the third with yellow copper and iron pyrites. The bedding of each jig can be obtained from the headings of the preceding jig. To obtain bedding for the first jig, put into the first sizing-screen a small section of larger screen. If the first screen is 6 mesh, put into the screen a strip of 4-mesh wire-cloth 6 inches wide, which will allow enough ore of the proper size for bedding to pass to the first jig, and keep the bed supplied. This will save the time and trouble of having to crush and screen ore to make this bedding.

By close attention to the bedding on the different compartments, a perfect separation of the different minerals can be made. If the headings of the first compartment show any copper, the bed is not heavy enough, and if the headings from the second compartment show galena, the bed on the first compartment is too heavy.

On the jigs use the least possible quantity of water to do the work, but be liberal with water on the revolving screens. The water in the jigs should be perfectly clear. If the jigs are muddy, slime is sticking on the ore, instead of being washed off in the revolving screen and going to its proper place on the buddle table.

Discussion.

Richard Pearce, Argo, Colo.: The subject of the concentration of low-grade ore is one of special importance to Colorado at the present time.

The State has hardly kept pace with other districts in that branch of mining industry. It has been customary for Colorado miners to sell their ores to the smelters in mass, just as it comes from the mine, but now the necessity for concentration is coming to be more plainly seen, especially in the low-grade ores of certain localities, not favored by cheap facilities for transportation to the smelters. It must be remembered, however, that in some instances concentration is accompanied with heavy loss in gold and silver. In the concentration of

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a mass of auriferous chalcopyrite in a gangue of quartz in California, only 40 per cent. of the gold was saved, the valuable mineral being finely distributed in the quartz. As an example of the opposite character, the London mine, between Leadville and Alma, may be cited, where, in concentrating from 4 tons to 1, there was a saving of over 90 per cent. of the gold. The gold in this case was contained in the pyrite.

CHARLES J. MOORE, Leadville, Colo.: The results attained by concentration at Leadville are not entirely satisfactory, and the work is not thought to be good unless 80 per cent. or more of the lead is saved. The ores subjected to concentration are chiefly sulphides—pyrite, galena, and blende. The silver occurs most generally in the galena and blende, not in the pyrite, except in those parts of the Leadville district where the ore occurs in fissure-veins in granite country, or in porphyry dykes, in which cases I have noticed the pyrite often to be the principal silver-vehicle, or, at least, equal to the blende as such, the galena taking the third place both in quantity and relative silver value. In these veins also is found considerable rhodonite or manganese silicate, but, as a rule, this mineral shows little affinity for silver; it is often, however, impregnated with pure pyrite and nearly pure galena, the former in the largest quantity, as we should expect from the frequent association of manganese and iron.

The percentage of saving of silver is less than that of lead, seldom amounting to more than 60 per cent. This is explained by the fact that, owing to the objectionable behavior of the blende in the furnaces, this mineral, carrying silver with it, is allowed to go into the tailings. The problem is, how to get rid of the blende and not lose the silver it contains; or how to save all the galena with the silver it carries, and not retain so much blende as to affect the cost of smelting unfavorably. If the crushing is extremely fine, silver is lost in the form of a light sulphide that floats on water.

At Aspen there is only one concentrating-mill, that at the Molly Gibson mine. The ores contain a large amount of barite, not easily separated from the ore. In some of the Aspen mines there is considerable lead, but the ores are generally dry. The gangue of the ore in Leadville is partly quartz and partly siliceous dolomite; in Aspen the gangue is principally a changed dolomite; in some mines, however, those which carry lead, the gangue is the changed blue limestone, a rock which carries from 80 to 96 per cent. of $CaCO_3$.

JOHN M. DESLOGE, St. Louis, Mo.: In connection with the sub-

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ject of the concentration of low-grade ores, something may perhaps be learned from an account of the German-American practice at the old mill of the Desloge Lead Company at Bonne Terre, Mo. The mill was destroyed by fire in March, 1886, and the mine is at present a part of the St. Joseph Lead Company's property. I must therefore depend considerably on my memory for the facts and figures I am able to give.

The ore yielded on an average about 7 per cent. of a non-argentiferous galena, and 1 per cent. or more of cobalt- and nickel-bearing pyrites. These minerals occurred in an almost non-siliceous, true dolomite, containing few or no fossils, and belonging geologically to the horizon of the Lower Silurian. The rock was impregnated unevenly throughout a stratum of many feet in thickness. Bodies of ore carrying 20 per cent. of galena were not unfrequently met with, but these richer portions were mixed with the leaner rock, for it was found that better results were obtained when the ore to be concentrated was kept at a nearly uniform standard. The ordinary products of concentration, aside from the waste, were two in number; one, locally called "mineral," containing about 70 per cent. of lead and 15 per cent. of dolomite, with little or no silica, and a middle product, known locally as "sulphide," consisting of about 33 per cent. of galena, 33 per cent. of dolomite, and 33 per cent. of pyrite, with something less than 1 per cent. of cobalt and nickel.

The ore came from the mine in ½-ton cars, and was dumped by cradles provided with balance-wheels, by means of which the empty cars were quietly and quickly righted. The ore fell in front of three 9″ by 15″ Blake crushers; originally two of them were of the lever pattern and the other eccentric, but, after long use, the eccentric not having two-thirds the capacity of one of the others, was replaced by one of a different pattern. In regard to the use of wrought-iron jaw-plates, I can say that my attention has been called to an idle crusher fitted with such plates, whose appearance as to usage and wear seems to indicate the practicability of employing that material.

The ore was broken so as to go through a 2-inch ring and thence directly to three pairs of 14″ by 30″ Cornish rolls, with chilled tires made of car-wheel iron. These usually wore from four to six months, but, at the time of the fire, there was one pair in fair condition that had been doing an average daily crushing of 80 to 100 tons a day of moderately hard limestone since the starting of the mill, about five years before.

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The coarse rolls delivered a product of hazel-nut size, which was taken by a belt-elevator in buckets holding 10 pounds each to three sets of 9 mm. screens, 3′ by 8′. All that was too coarse to pass through the screens returned to the rolls for recrushing. The ore that passed through fell upon three sets of 7 mm. screens, 3′ by 6′. The screen-openings were all round-punched.

The sized ore between 9 mm. and 7 mm. went to three pairs of 2-sieved roughing-jigs with lever slot-motion, the speed being in proportion to the arc travelled. The stroke was $\frac{7}{8}$ inch, with 110 revolutions a minute. The sieve-wire was No. 8. A Lake Superior Collom roughing-jig, was, after some service, abandoned, though its work compared favorably with that of the other jigs.

The ore that passed through the 7 mm. screens went to water-current classifiers, one for each screen. Here three classes were made: one from 7 mm. to 5 mm.; a second, from 5 mm. to 3 mm.; and a third below 3 mm. The ore of size between 7 mm. and 5 mm. went to twelve roughing-jigs, of the same pattern as those previously mentioned, but with No. 15 sieve-wire instead of No. 8. The stroke was $\frac{5}{8}$ inch, with 120 revolutions a minute. The ore between 5 mm. and 3 mm. went to Hartz, or eccentric-jigs. Of these there were two sets of four each with three sieves of No. 15 and No. 20 wire. The stroke was $\frac{1}{2}$ inch, with from 150 to 200 revolutions a minute, according to the ore.

All the jigs in this portion of the mill had the automatic cap-discharge from the sieve. The roughing-jigs had, approximately, a capacity for treating 25 tons in twenty-four hours; the eccentric-jigs a capacity of 12½ tons. The product known as "mineral" was obtained from the first compartment of the jigs, and generally, also, from the first hutch-boxes. The product from the second compartment of the roughing-jigs was usually recrushed, but that from the second compartment of the eccentric-jigs working on the stuff between 5 mm. and 3 mm. constituted the "sulphides." The product of the third compartment was recrushed and subsequently treated in another part of the mill.

The two sets of Cornish rolls for recrushing were of the same pattern as those previously mentioned, but they were run at about double the speed, making 25 revolutions a minute and crushing 40 tons, more or less, in twenty-four hours. The product of these rolls went to a 3 mm. screen. The part that was too coarse to pass through the screen was returned for recrushing. That which passed through went to two sets, four in a set, of 3-sieve eccentric-jigs, from

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which "sulphide" and some "mineral" were obtained. The tailings or "chats" went to settling-tanks and were thence delivered into side-dumping cars and hauled away by a locomotive running on a track of 3-foot gauge. The slimes and tailings were handled in the old mill, where there were a set of rolls, a screen, two roughing- and two eccentric-jigs, two Evans and one Coggin table.

These tables were adopted after an exhaustive competition with Rittinger side-bump tables. Their capacity was at least five times that of the Rittinger tables, and they gave nearly as clean a product; they required less power and less attention, used no more water, and showed less wear. The tables were used for treating slimes from the settling-tanks. Each table could treat about one ton an hour, making 60 to 80 revolutions an hour. One table was used, with fair results, to treat the waste slimes of an old settling-pond. The headings of the tables were clean, and the middlings went to light-bumping tables or vanners for higher concentration.

The rated capacity of the mill was 300 tons of mine-rock in twenty-four hours, and the average work was 250 tons. The water used amounted to 1200 gallons a minute. The mill gave employment to about 33 men, including those needed for loading the tailing-cars, but not the engineers or firemen, who were engaged also for a part of their time on mine-work. The approximate cost of milling was 40 cents a ton.

The breakers required the service of about 14 men. From my knowledge of the Gates crushers I am of the opinion that, for the same capacity, they would not need more than one-quarter the number.

The rolls for recrushing the finer material did not give satisfaction. The loss was about 30 per cent. on a 10 per cent. galena. This loss was due to the presence of dolomite, to which particles of galena adhered, or of fine galena which was washed away in the tailings. There was great need of pulverizers and slime concentrators that would do good work and have large capacity. Several pulverizers were tried but without success. A single-head steam stamp-mill, constructed on the principles of the Rand drill, was tried, but that failed.

For fine grinding, the Heberle mill, used by those who adopt the Lake Superior practice, and, for slime concentration, buddles of the Evans pattern, with light-striking tables to take the buddle middlings, seem to me worthy of consideration where such losses like those at Bonne Terre occur. The old mill was started with 15 mm.

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screens; but in the other mill, where it was planned to discard down to 7 mm., less attention was given to screen-sizing, and more to the work of the roughing jigs.

In milling our extensive deposits of low-grade ore with a low price of lead, it was commercially important for us to decide on the most economical practice, and to determine whether it were better to treat rapidly a large tonnage, even at a loss of 30 per cent. or more, or to work with a higher percentage of saving on smaller quantities. In the former case, one screen-sizing may be the best, as is the present practice of the St. Joseph Lead Company; but, in my opinion, what these ores require is two screen-sizings, the first to not over 7 mm., and the second to some finer size (unless the screen is replaced by some other modification of sizing apparatus), and good grinding and sliming machinery.

THE IRON RESOURCES OF COLORADO.

BY PROF. REGIS CHAUVENET, GOLDEN, COLO.

(Colorado Meeting, June, 1889.)

I HAVE been requested to sketch the iron resources of the State of Colorado, with reference both to existing, *i.e.*, working mines, and to prospects more or less developed, but not contributing anything as yet to actual shipments. I should mention, though disliking apologies, that there is one region of the State which I believe to be important, and which has been assigned to me (with reference to its iron-deposits particularly) in the field-work of the present year for the School of Mines, which has not been systematically explored, and whose data are not in such shape as to enable me to give them in this paper. The region in question is Pitkin county and part of the adjoining country.

Regarding the geological positions of the veins or deposits, they are sufficiently varied. In Boulder county there is a well-defined vein of titaniferous magnetite in ancient granite or gneiss, while in Park county there may be seen a deposit now under construction, as will later be described; when finished it will belong, let us say, to the post-quaternary. Between these, in Cambrian, Silurian, and Lower Carboniferous formations, are various deposits, and even the

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Cretaceous shows some concretionary ores, though of minor importance. The genesis of these deposits is too large a subject, and too far out of the line of the present brief summary, for me to do more than allude to it.

In presenting the subject, the ores will first be taken up by counties, and the present and prospective centers of iron manufacture be afterwards named.

Beginning with Boulder county, we have the vein in Archæan rock already mentioned, which is found on Caribou Hill, close to many rich silver veins, though not itself forming part of the system of veins which carry the precious metals. This vein powerfully affects the compass in its vicinity, and is called a deposit of magnetite, though, so far as its analysis is concerned, it might be named ilmenite, since it contains the unusual amount of 36 per cent. of titanic acid. Its percentage of iron is from 36 to 38. The ore is compact, jet black, and, of course, worthless as iron-ore. It is alluded to, not as forming any part of the "iron resources" of the State, but to call attention to the wide diffusion of titanic acid in Colorado, this being only one of four deposits, in as many counties, which must be thrown out of account as iron mines, owing to the presence in large quantities of this pestiferous enemy of the iron smelter.

Historically, Boulder county has the honor of being the first to produce pig-iron, not only in Colorado, but probably in the whole Rocky Mountain region. The iron was not made, however, from ore of the kind mentioned above, but from concretionary boulder ores, of the Cretaceous coal strata, found near the city of Boulder. It was made in a charcoal furnace. Some description of the work doné is given in Hayden's reports on the geological survey of the territories for 1867, 1868, and 1869. These ores, still to be found, though not in promising quantities, are poor and very calcareous. It appears, however, that the founder of the furnace and the compiler of the United States statistics, were at fault between them, since we read on the one hand that the ore contained 70 per cent. of metallic iron, an amazing statement to be found in a United States publication, while on the other it is stated that it required three tons of the ore to produce one ton of pig-iron. It must have been a very long or a very short ton, respectively. I shall not attempt to decide now whether the arithmetic or the metallurgy was most at fault. This region is mentioned only in deference to that custom which prescribes a little history at the outset of a sketch of the present. The furnace, the date of whose erection I cannot give with precision, has long

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been abandoned, and it is not probable that the manufacture of iron will ever be resumed in this section.

Coming southward, we find our attention arrested in the north part of Park county, by the phenomenon of a bed of limonite now in active process of formation. This, indeed, may be seen elsewhere, but it may be doubted whether there is another place where the whole "genesis" of bog-ore can be seen so beautifully. Before proceeding, then, to describe the working deposits of the State (for this cannot be considered to be one of them, as yet), it may be worth while to describe this, as a metamorphosis of some interest, geologically, if not industrially.

The South Platte river has a number of forks, and it is on the North fork, or the streamlets which unite to form it, that the beds in question are now forming. In two little valleys, known as Hall's valley, and Handcart gulch, we find an abundant supply of bog-ore, its character being "mixed," however, in the present stage of formation. The bed in Handcart is the more typical of the two. We find, as we ascend the gulch, that the stream is so strongly ferruginous, from the ferrous sulphate dissolved in it, that its inky taste is quite disagreeable. As we trace it to its head-waters, which are close to the main divide of the continent, we see the rocks above blazing in red and yellow, from the oxidation of the chalybeate water, which percolates through them, charged with the sulphate, the result of the solution of pyrites, great beds of which are known to exist in the region. Descending into the lower part of its narrow valley, the stream runs into what is almost a marsh, with dense vegetation, and here it deposits its iron so completely, forming the still growing beds of limonite, that iron can with difficulty be distinguished by the chemist in the issuing water. Every stage of the ore-formation can be traced, or, one might say, seen. Tufts of bog vegetation can be pulled up from a mush of half formed oxide, the roots crusted with incipient ore, while from the more solid portions, nearer the present bed of the creek, true ore can be found in considerable quantities. Some of this has been analyzed by the writer with the following result:

SiO_2,	7.13
Fe_2O_3,	67.55
P_2O_5,	1.82
CaO,	1.22
H_2O and organic matter,	22.37—Total, 100.09

Here, then, is a bed of ore in actual process of construction. It

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would be possible to-day to extract from it much ore fit for smelting, and it can hardly be doubted that in a future age it will present all the features of a true solid limonite deposit.

Dropping these curiosities of formation, however, we come to deposits more worthy of being classed as belonging to our "resources." First taking Chaffee county, we have the Calumet mine (magnetite), which is an interesting, but by no means uncommon, type of the oxidation of a great mass of pyrites. The vein lies in a singular rock, apparently a conglomerate of feldspar and hornblende, said to belong to the Lower Silurian. It has been developed on quite a large scale, both by shaft and adit, and is in places nearly forty feet in width, with exploration to about four hundred feet in depth, and much more than that in length. Its worked-out stopes are a wilderness of massive square-set timbers, indicating that many thousand tons have found their way to the furnaces of the Colorado Coal and Iron Company, for the vein forms part of the property of that corporation. It is true Bessemer ore, and here must be pointed out a characteristic of this and other magnetite mines of the State, viz.: their unusual freedom from phosphorus. It is this which has made this vein so valuable to the steel-works at Pueblo. Having analyzed samples of this ore at various times, and having also been kindly furnished with copies of many analyses from the laboratory of the Company, I am able to state that it will not average over 0.008 per cent. of phosphorus. The smallness of this figure often excites a smile of incredulity from iron men, but it has been amply verified.

The ore, however, is not so free from sulphur, showing from $\frac{1}{10}$ up to as high as 2 per cent. in shipping lots, perhaps yielding something under 1 per cent. in a general average. Its average percentage of metallic iron (actual shipments) is 57 (lowest, 50; highest, 63). It is estimated by the Company that they have here a supply which may be reckoned upon for years. To such of the members of the Institute as may desire an exciting ride, I recommend the railroad trip from the mine to Salida, said to be over the steepest grade in the country. Very few locomotive engineers are entrusted with the loaded ore-train in making this terrific descent, in which, it need hardly be said, the steam is used as a hold-back, not as a propeller. The trip to the mine and return from Salida may easily be accomplished in a day, as the distance is only 10 miles, that is, 10 miles horizontally; the vertical distance will be estimated by those making the journey at a not dissimilar figure.

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Only brief mention can be given the other working mines of the Colorado Coal and Iron Company. The works themselves are too well known to call for any treatment at my hands.

In Fremont county is the mine known as the Grape Creek. The ore is titaniferous, showing a very steady average of about 14 per cent. of titanic acid. Its iron runs about 48 per cent. It is no longer used in the blast-furnace, but is useful as "fix" in the puddling-mill. The deposit is a large one, and it is a pity that its character prevents its more extensive use.

In Saguache county we have the Hot Springs mine, an irregular mass of limonite, from which considerably over 100,000 tons have been taken, and which is considered good for much more. The average anaylsis of this ore (taken from a great number of car samples), omitting minor constituents, gives about 43 per cent. of iron, 20 per cent. of silica, and 13 per cent. of water, with very small amounts of other matters, including, curiously enough, barium, but it is for its very low percentage of phosphorus that it is remarkable. This element is generally supposed to exist as a matter of course in most limonites, and the steel-maker regards such ore as quite thrown out of account for Bessemer use. But here we have a limonite with only 0.03 per cent. of phosphorus in average ore, and as the ore is found to work well in the furnace, it has remained a favorite with the company in spite of its high silica. The mine is connected with the main line of the Denver and Rio Grande Railroad, as is also the Calumet. It lies in the extreme northern end of San Luis Park, about 10 miles from the village of Villa Grove.

Leadville has become famous as a silver and lead region, but its iron-deposits should not be overlooked. The Breece and Hull mines are the only ones we need dwell upon. Both of these have contributed to the stock at the Pueblo works, and the Breece has sent out thousands of tons for the use of lead smelters as flux.

The Breece ore has been described as a magnetite, though it appears that it now contains more hematite. Geologically it may be said to be in the Lower Carboniferous; it occurs at the contact of two sheets of porphyry, underlain by the white porphyry so well known in the district. This ore has been developed to a width of over 40 feet, but its limits are unknown. In the Monographs of the United States Geological Survey, vol. xii. (Emmons) will be found the wonderfully elaborate analysis of this ore by Dr. Hillebrand, taken from a number of heaps at various smelters. By this it appears that the ore carries besides iron, manganese, nickel, cobalt,

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zinc, copper, gold, silver, arsenic, antimony, calcium, magnesium, aluminum, titanium, carbonic acid, phosphoric acid, chlorine, and water. Such an analysis, presented merely as qualitative, might well terrify an iron-master. He would be somewhat reassured upon learning that the sum total of all the impurities, silica excluded, would not amount to 1 per cent., of which over one-half is magnesia. From a long series of analyses sent to me by the Colorado Coal and Iron Company, I take the average of 60.5 per cent. of iron, 5.80 of silica, and 0.03 of phosphorus, showing this to be the richest iron-ore in the State in average shipping value. The main use of the ore has always been the furnishing of flux to the smelters, and many a ton of its iron now lies irreclaimable in the slag dumps of Leadville.

Of the Hull mine of Leadville I can give little account, except that it is used alone in the furnace at Pueblo, yielding a spiegel-iron of superior grade, with from 22 to 24 per cent. of manganese. Its analysis shows 32 per cent. of iron, 17 of manganese, 13 of water, and about 5 of silica. It is worth noticing that the State already yields every product for the manufacture of Bessemer steel.

This ends the list of producing mines. We now cross the range, and in Gunnison county find several deposits which are in their infancy as to development, but which, taken in connection with the coal-fields of the same region, give hope of the establishment of an iron industry in the mountains; that is to say, in the mountains proper, as distinguished from the country at their feet. The largest veins here, so far as known, are magnetic, probably derived from pyrites, as were those in Chaffee and Pitkin counties. It should be mentioned that systematic exploration for iron and manganese is now going on in Gunnison county, and that new discoveries are claimed to have been lately made, but no account of these can be given in this paper.

The largest vein in Gunnison, so far as known, is the Iron King, close to the mining village of White Pine, 15 miles from the nearest rails of the Denver and Rio Grande Railroad. It is all down grade to the rails, however, and thence to Gunnison City. While only about a thousand tons have been taken from this prospect, to be used as flux at a Gunnison lead-smelter, it already appears as a great deposit. Its width is 70 feet, where cut, not all solid ore, but very largely so, while its croppings extend to such a distance that, if they can be held to indicate everywhere a similar mass of ore below, the figures representing the total amount

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would rise into millions of tons. Without any exercise of fancy, however, this can be seen to be a fine vein, if a vein it is. It seems to lie between quartzite and limestone, and from various parts of it can be taken ore in mass, running over 60 per cent. in metallic iron; I have, personally, taken samples giving 67 per cent. So far as exposed, this mass can be mined so as to ship ore at 60 per cent., or very near that figure. Its more solid portion lies nearest to the limestone wall. Rails have not been laid to White Pine, though, for most of the way, the grade would not be severe. A general sample, which included much material rejected in ordinarily careful mining, yielded 58.75 per cent. of iron, less than 4 of silica, and only 0.04 of phosphorus. It is Bessemer ore, like that of the Calumet or Breece mines. A brother to it lies on Gold Hill, in a different, though not very distant, part of the county. This is known as the Cumberland mine. It may be as large as the Iron King, but is not as accessible, and has not been opened to the same extent. But it is a massive deposit, and its analysis is as good or better than that of the latter. The Cumberland lies at a high altitude, and about 10 miles distant from the nearest railroad (the Union Pacific).

Next, as a curiosity, not as a commercially valuable mine, I would name the Cebolla Creek titanium deposit. No one knows how the ore will hold out here, but as it can be developed in many places for some miles, it will last, in all likelihood, far longer than there is any use for it. It contains from 9 to 36 per cent. of titanic acid. We have all read in our text-books on chemistry, that titanium is a "rare" metal. I do not think I risk much in saying that this is an inexhaustible stock of it.

In the same district are many small veins of rich manganese-ore, whose localities I will not name in detail. Whether they will be found to be worth working, it is too early to say, though, in another part of the county, there is already a deposit of workable magnitude, which can be made the basis of spiegel manufacture. This is on Taylor River, 27 miles north of Gunnison City. It is, or seems to be, a great bedded mass, having been found at points 1500 feet distant from one another. Containing almost no silica, its analysis is that of a carbonate, which near the surface has been partly oxidized. After roasting, this ore yields, iron, 34.5 per cent. and manganese, 13.6 per cent., a beautiful proportion, while it is rich in lime and magnesia, so that no fluxing material, except silica, would have to be added. This should yield a spiegel similar to that now made at Pueblo. Omitting mention of many other iron-fields,

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I will pass to the general fitness of this county for iron industries. When we have iron-ore, coking-coal and limestone, all within easy transportation distance of the same point, and all to be carried "down grade" to it, there would seem to be a fair prospect for an iron center. The coal-fields of Gunnison county are already in a high state of development, and two or three branch lines run to them from Gunnison City. At this point iron should be made for $12 a ton, or even less than that. We now have one iron center at Pueblo, and it is probable that our next will be somewhere on the western slope of the mountains. We all remember the story of what the Texan said his State lacked. Gunnison City lacks only capital and a market. But the West of to-day is the East of to-morrow. Some of us have seen iron made where the prophets of only twenty years ago said it could never be made. We have seen it competing from points considered outside of the limits of competition ten years before the erection of the furnaces. We should beware then of laughing at the future of any new field. It may be that before the American Institute of Mining Engineers again meets in Denver, Colorado will be exporting her pig-iron to the Pacific slope.

NOTES ON THE GEOLOGY OF THE ASPEN MINING DISTRICT.

BY W. E. NEWBERRY, ASPEN, COLO.

(Colorado Meeting, June, 1889.)

THE Aspen Mining District is situated at the eastern edge of the system of stratified rocks of western Colorado, where this edge meets the ridge of metamorphic granite which divides the State into two unequal parts. In following the valley of the Roaring Fork westward from the continental divide towards Aspen, one passes through a country of granite and gneiss, with no sedimentary rocks until the town is nearly reached. At this point the Cambrian quartzites first make their appearance, lying on the granites, with a dip of from 30° to 45° to the northwest. Beyond these quartzites and conformably upon them lie dolomites, limestones, shales, porphyries and sandstones, gradually flattening in general dip as one proceeds westward, but much disturbed by local foldings and faultings.

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More in detail, the succession of these strata upwards from the granite is as follows, the measurements being an average of a number made by Mr. D. W. Brunton:

	Thickness.
1. Cambrian quartzite,	400 feet.
2. Silurian quartzites and limestones,	460 "
3. Lower Carboniferous dolomite,	225 "
4. Lower Carboniferous blue limestone,	110 "
5. Shales and shaly limestones of the Middle Carboniferous,	50 to 450 "
6. An intrusion of diorite, maximum thickness,	400 "
7. Middle Carboniferous limestone,	10 to 160 "
8. Jura-triassic sandstone, with a thickness of many hundred feet.	

The characteristics of these strata are as follows:

The Cambrian quartzite is compact and generally fine grained, and from white to pink in color. The Silurian strata consist of numerous beds of impure limestone, with bands of quartzite. The limestones are generally light colored, while the quartzites are coarse grained, and from red to white. In this series is a band of lithographic stone about two feet thick.

The Lower Carboniferous dolomite, or brown limestone, is dark colored throughout. Near the surface, and in the neighborhood of the contact with the overlying blue limestone, and near fault-planes, in short, wherever exposed to the action of atmospheric water, it is of a reddish-brown color, due to the oxidation of the iron contained in it, and it breaks up into small cubical fragments, from which property it derives its local name of "short lime." Where it has been less subjected to decomposition it is more massive, and almost black in color. It contains from 20 to 38 per cent. of $MgCO_3$, and sometimes as much as 5 per cent. of oxide of iron. It contains nodules and lenticular bodies of chert, and, in some portions of the beds, masses of blue limestone or calcite of considerable size.

The Lower Carboniferous blue limestone is massive, without distinct cleavage, except where it is crushed by movement, when it shows something of the short structure of the above mentioned dolomite. The color is a bluish gray, and it is in some cases coarsely crystallized, and in nearly all cases shows some crystallization. In the neighborhood of the dolomite, and particularly near an ore-body, the blue limestone becomes stained with iron, and, with the cleavage it has under these conditions, it is often difficult to distinguish it from the dolomite without analysis. The crystallization is the only

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safe guide, a freshly broken surface of the blue limestone showing flat, scale-like crystals, while the crystals of dolomite are like fine needle-points, without perceptible area. In the blue limestone stratum are a number of sheets of dolomite, similar in appearance to the main body of dolomite below. From the fact that these sheets are often lenticular in shape, and cross the bedding of the blue limestone to connect with each other, and from other facts in the same connection, there is reason to believe that the beds of blue limestone and dolomite were originally deposited as a homogeneous body of limestone, and that the dolomitization of the lower portion and of the sheets in the upper portion was subsequent to the deposition of the rocks. The blue limestone contains from 92 to 99 per cent. of $CaCO_3$.

The shales and limestones lying above the blue lime are highly carbonaceous, in places streaks of coal being found of sufficient purity to be burned in a blacksmith's forge. These beds are also highly charged with iron pyrites.

Intruded into these beds, separated from the blue limestone by from 5 to 100 feet of shale, is a great mass of diorite. This rock, locally called porphyry, is light in color, with bird's eyes of light blue, and small crystals of pyrite. The porphyry forms the swell in Aspen Mountain, directly opposite the town. It is much decomposed near the surface, and has not been identified on Smuggler Mountain, across the valley from Aspen Mountain. It extends a long distance to the southward, being shown at least as far as Ashcroft, 12 miles from Aspen.

The above described strata are cut by the valley of the Roaring Fork nearly at right angles to their strike, and therefore cross under the town, and appear in regular order on either side of the valley. The geological structure of Aspen Mountain is, however, much more complex than that of Smuggler Mountain, to the north. A vertical section of Aspen Mountain, taken on a northwest and southeast line, through what appears from the town to be the summit of the mountain, would show three occurrences of the strata above mentioned, resembling the three slopes of an S-fold, with the syncline at the eastern and the anticline at the western end of Aspen Mountain, and this folding, complicated by faults at the base of the syncline and at the crest of the anticline, is the cause of this reappearance of the strata.

The bottom of the synclinal fold has a strong dip to the north, forming a basin, concave towards the city.

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The prime cause of this folding has been a great granite upheaval, which has lifted the strata at the west end of the mountain, and compressed into the form of a basin those lying to the east. The strata to the west of the granite upheaval were unable to stand the strain, and broke, forming a great fault, the line of which is now marked by the course of Castle Creek, which flows into the Roaring Fork to the west of the town.

The series of strata on the north side of the valley seems to be much less disturbed than that on the south, and the rocks follow their regular order, from east to west, without important break as far as observed.

The principal ore-bodies of the district occur in the Lower Carboniferous strata, including the dolomite and blue limestone above mentioned. The ore occurs generally in the neighborhood of, or on the contact between, these two strata. The fact that the strata are cut by numerous faults, making generally an acute angle with the direction of the dip, and more or less nearly parallel to the strike, complicates matters, and makes two classes of contacts between these rocks, one being the stratified contact, with the blue limestone above and the dolomite below, and the other the faulted contact, with the rocks in the same relative position when the fault is of the ordinary kind, and in the contrary position, *i.e.*, with the dolomite above and the blue lime below, in the case of a reverse fault. The throw of these faults is in some cases so great that the shale is brought opposite the dolomite, and a faulted contact formed between these rocks.

The stratified contact, having the general dip of the formation, inclines at an angle of from 10° to 45° to the northwest, the general strike of the formation being about north 45° east. There is no evidence of any great motion upon one another of the rocks forming this contact, large areas of the contact being of a tight or welded nature. In many places, however, there is evidence that the rocks have been parted, at or near the contact, probably by folding, and that the blue limestone has moved on the dolomite sufficiently to cause an increase in the size of the openings, by the correspondence of the ridges and hollows in one with the hollows and ridges in the other.

Thus a portion of the ore found in the neighborhood of the stratified contact consists of foreign material, mixed with fragments of the surrounding country-rock. But a great part of the ore, and I think I am safe in saying, the greater part of the ore shipped from the

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camp, consists of an impregnation or mineralization of the country-rock in place, either the blue limestone or the dolomite, in the neighborhood of the stratified or faulted contact.

The method by which the mineralization has been produced differs in the cases of the two rocks. The dolomite does not seem to have been more readily decomposed by the mineral-bearing solutions, but owing to its short structure, and many cleavage planes, it has been more readily permeable by them, so that much of the brown limestone ore consists of small cubes of nearly unaltered dolomite, with the valuable mineral deposited mainly in its cleavage-planes. The dolomite has, however, in many cases, thoroughly lost its structure, and is often reduced to the condition of a dolomite sand, impregnated with valuable mineral. The mineralization of the blue limestone seems to have been accomplished by the replacement of a portion of the calcite, without materially altering its structure. In many cases the rock retains its appearance, and the mineralization can only be discovered by assay. Even in such cases the rock usually appears more crystalline and porous than when barren.

The faulted contact is characterized by a steeper pitch than the stratified contact, and by the great quantity of wall-rock contained by it, and the strong evidence of motion. The welding of the walls noted above is, of course, absent.

There seems to be a general system of ore-chutes in these limestones, having a southerly direction, often interrupted by faults, but probably in many cases continuous along the fault-planes. From the developments so far made, the trend of these ore-chutes seems to be about south 65° west, swinging slightly to the south with increased depth.

The gangue of the Aspen ores is generally either limestone or sulphate of baryta, or both, passing from ores like some of those from the Aspen mine, containing as high as 90 per cent. of carbonate of lime, to others from various mines containing as a maximum 70 per cent. of sulphate of baryta. The amount of silica in the ores is generally small, rarely running over 20 per cent. The bulk of the ore is very low in lead, that containing 25 per cent. being a rarity. An average of the ores at present shipped from the district would contain not far from 55 ounces of silver per ton, with less than 5 per cent. of lead, and from 12 to 15 per cent. of sulphate of baryta. An average smelting charge of about $9.00 per ton is paid on these ores, delivered in Denver, which with the railroad freight of $8.00 makes a total average expense of $17.00 per ton after delivering on the

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ears. Very little ore is shipped containing less than 25 ounces of silver per ton.

In conclusion, the Aspen mineral-bearing formation has been proved to be productive from Tourtelotte Park on the south to the St. Joe mine on the north, a distance of over 4 miles. The same series of strata has been identified uninterruptedly to the south for a distance of about 12 miles, and to the north for 26 miles. For this whole distance prospecting is being actively carried on with indications which justify the expectation of fully as good results in other portions of the belt as have been obtained in the neighborhood of this city. The whole distance of 4 miles, above mentioned, is one chain of producing mines, except where interrupted by the valley of the Roaring Fork. Here the great depth of the débris, and the quantity of water have prevented the prospecting of the ore-bearing formation, but there is every reason to believe that the ore-bodies are continuous under the city.

THE SUDBURY ORE-DEPOSITS.

BY E. D. PETERS, JR., SUDBURY, ONTARIO.

(Ottawa Meeting, October, 1889.)

THE Sudbury ore-deposits possess a peculiar interest for a variety of reasons. In the first place, they are deep within the borders of the Huronian rocks, and are consequently amongst the oldest deposits that we know of, unless, indeed, the ores were deposited at a much later period than the country rock. Again, they carry nickel in unusual proportions, and lastly, they are unique in containing a small amount of platinum, although arsenic is not present.

I shall not pretend to go into the geology of this region. The Huronian and Laurentian rocks are familiar to all American geologists, and these present only the usual series of gneiss, graywacke, quartzites, greenstones, clay-slates, etc., characteristic of this gigantic system of rocks, which has excited so much discussion of late years.

As in all parts of Canada where it occurs, I believe, the rocks of this system are tilted to such an extraordinary degree that it may be said that the whole country is standing on edge at an angle of something like 70°. The general strike of these rocks is southwest

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and northeast, and as the ore-bodies follow the lines of stratification, if it can be called stratification, it follows that the course of the latter is the same as that of the rocks, although local twistings and faults are so frequent that no absolute rule can be laid down as to their direction.

But, at least, one general feature can be stated as universal; and that is, that all the ore-bodies yet discovered in this region are in close proximity to somewhat extensive dikes, or tilted beds, of diorite. It is usually the case that the ore occurs at the point of contact between diorite and the graywacke, although in some cases it is found in the midst of the diorite itself. The latter is, of course, a volcanic rock, and is composed properly of quartz, feldspar and a considerable proportion of hornblende; but by the predominance of one or the other of these constituents, and the corresponding diminution of one or more of the remaining minerals, it forms almost every variety of rock imaginable, and differs so greatly in appearance as to deceive any one who has not had an opportunity to observe its modifications.

This diorite forms the gangue-rock of our ore-deposits to a great extent, and is a rather favorable constituent than otherwise, as its varied bases and comparatively low percentage of silica makes it far less refractory in the furnace than the quartzose vein-matter that so commonly accompanies copper-ores.

The ore itself is a mixture of very pure chalcopyrite and magnetic iron-pyrites or pyrrhotite, carrying a considerable amount of nickel, this metal no doubt replacing an equivalent amount of iron.

The mines were worked originally for copper, and it was not until a shipment had been sent to be smelted that it was discovered that they were nickel-bearing to an extent that rendered them far more valuable for that metal than for copper.

As the principal amount of work at Sudbury has been done by the Canadian Copper Company, and as I am naturally most familiar with its mines, I shall describe them more particularly, since in so doing I shall be describing the whole district correctly.

I know of no mines there that differ materially from the mines of the Canadian Copper Company, except that most of the later-discovered deposits have shown neither the size nor the richness of the earlier-discovered ones, a fact that is notorious in the history of almost every mining camp with which I am familiar.

It is very difficult to give any correct statements as to either the size or richness of these beds, since they are so variable in the former

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particular as to make it impossible to give an average figure, while in the latter it depends upon the class of deposits to which we are referring. In order to give some slight idea as to what these mines amount to, I will divide them roughly into two classes: first, those which are composed of extremely massive pyrrhotite, and are of enormous extent, so large, in fact, that we have as yet obtained no idea of their boundaries; and second, those which are more rocky in their nature and less extended in size, but very much richer in both copper and nickel. I do not know but that in justice I ought to make a third class, to which one of our most important mines belongs. They are not only about as extensive as the first class, but are nearly as rich in valuable metals as the second series. This is a most unusual and pleasing combination, but is certainly attained by the Evans mine, belonging to the Canadian Copper Company, and quite possibly by one or more of its yet undeveloped deposits.

The main peculiarities of each of these three classes of mines may be illustrated in describing the three mines which the Canadian Copper Company is now working most extensively for ore, and of which, curiously, each belongs to a different class.

As an example of the first class, I will cite the Stobie mine, situated about four miles north of Sudbury, on a branch of the Canadian Pacific Railway, built expressly for the use of this mine. Before being opened at all, it simply appeared to be an immense rounded hill of red gossan. As to the length, I will only say that, although we have opened it but a few hundred feet longitudinally, yet the outcrop shows it to continue in a more or less unbroken condition for some miles. Upon removing this gossan, which consists of a brown iron-ore in regular stratified layers (the product of the decomposition of the pyrrhotite), we come, within 2 to 6 feet of the surface, upon the unaltered pyrrhotite in a massive condition. At intervals limited bands of rock occur, and occasionally considerable horses of diorite or of mixed ore and rock; but as a rule the ore is massive pyrrhotite, with occasional pockets of very pure chalcopyrite, and not infrequently rounded and even angular occluded masses of diorite, from the size of a chestnut up to boulders weighing many tons.

The position of the ground is favorable for open-cast work, and we are at present simply taking the whole hill down as we go, on a level with the valley where we start. We have in some places already cut over 100 feet across the ore, thus proving the width to be very great, and also obtaining a fine face for blasting down the

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ore in great quantities. Our practice is to bore by air-drills a series of eight to ten holes, about 10 feet deep, and as far back from the face of the cliff as we think it is safe to go. The holes are loaded heavily with dualin, and fired at once by a small dynamo, operated by hand, in the usual way. In this manner we throw down several hundred tons of ore at a single blast, and for two or three days the men are kept busy in block-holing the large fragments and breaking up and loading the ore, preparatory to a fresh blast. Of course, there is nothing original about this practice; but it is rather unusual as applied to nickel-ore.

This ore, although lower in both copper and nickel than the ore from the other two mines, is yet rich enough in both metals to compare with the best Norwegian or German nickel-ores, and is especially valuable to us locally on account of its high tenor in iron. The latter, after roasting to remove the sulphur with which it is combined, makes a most welcome flux for our richer, but more rocky, ores from the other mines. Besides cutting across the vein for a considerable distance, as already mentioned, two tunnels have been continued still further across it for some 60 feet, showing massive ores in their entire extent as well as in their extreme face. We hope and intend to extract a large portion of our winter's supply of ore from these two tunnels, which will enable us to work in comfort when the weather out of doors would be too bad. Our work at present is not economical, the ore being handled mostly in barrows, and being spalled down to size by hand. This is only a temporary make-shift until a new crusher comes to take the place of an old one moved elsewhere. Then the ore will be handled directly from the face of the quarry by a derrick operated by steam-power, and conveyed in cars directly to the breaker, where it will be crushed and screened by machinery and delivered automatically into the small cars, which will run directly over the large railroad cars and dump the ore into them very cheaply and conveniently. Within a fortnight this will all be in operation, and the hand-work entirely discarded.

The second type of deposit is well illustrated by the Canadian Copper Company's Copper Cliff mine. This is situated about four miles west of Sudbury, and some six miles southwest of the Stobie mine, in a bee-line. The ore consists of the same pyrrhotite and chalcopyrite, but by no means so massive, or in such an enormous deposit. There are many places in the Copper Cliff mine where the ore is massive, but in other parts, although rich in the valuable metals,

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it is much intermixed with diorite. The ore occurs here in irregular masses of several thousand tons each, situated apparently between two cleavage-planes in the country-rock, so that we can usually find new deposits by drifting. Although there are absolutely no stringers of ore or veinlets of quartz to connect these ore-bodies, the country-rock in their vicinity is usually speckled to a greater or less extent with ore, and this is often the only indication of our immediate proximity to a large ore-body.

At present, the Copper Cliff shaft has reached the 500-foot level on an incline, and the ore at the greatest depth yet discovered retains its full value in nickel and copper. As the pitch of the shaft is somewhat less than that of the ore, it has long since left the vein, and is now at some distance above the hanging-wall of the latter, so that we have to drift some distance to strike the proper cleavage-plane, where we may expect the ore. In the fourth level we have struck a fine deposit of ore, while the fifth level is just fairly started. The third level is our active level at present, though we hope before long to have the fourth and fifth levels in even better shape than the third.

The pure sulphide of nickel, millerite, is found in many mines which contain no nickel in paying quantities, while in our mines it has only occurred once in the Copper Cliff mine. Then it was in very small quantities, and presented the characteristic appearance of fine wires.

The Copper Cliff mine is well equipped with machinery. It has a double skip-road on an incline way to the bottom of the shaft, the skips dumping automatically at the mouth of the breaker in the top of the rock-house. Here the ore is sledged to a proper size for the 15-by-9 in. Blake breaker, which has a capacity of about 20 tons an hour, and passed through a revolving screen, where it is sized into three classes, suitable for the succeeding operation of heap-roasting. The coarse size passes a 4-inch ring, the medium size, or ragging, about passes a $1\frac{1}{4}$ ring, while the fines go through a circular hole of $\frac{1}{4}$-inch diameter. Each of these sizes falls into a separate bin, under which a car runs on T-rails. Thus the ore is loaded automatically into cars holding $1\frac{1}{2}$ tons, whence it is transported to the upper story of the ore-shed. There it falls into a series of bins, from which it is loaded by means of inclined steel shutes into the railroad-cars, and goes direct to the roast-heaps. Aside from the rock-breaker and its engine, this mine has a fine double-cylinder hoisting-engine with two drums, and a 7-drill air-com-

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pressor of the Ingersoll Company's make. We do nearly all of our drilling with Ingersoll drills, using compressed air, and on the whole find them efficient and tolerably economical. The further equipment of the mine consists of a machine-shop, just about completed. It contains a 20-foot lathe, a good sized planer, as well as a drill-press and an Acme bolt-cutter, so that we expect soon to do all our machine-work at home, thus effecting a large saving. These tools are driven by a 10-inch vertical engine, taking its steam from the main boilers in the compressor-building. We further have a couple of home-made hydraulic piston-jigs, which we are about to use on some of our fine ore that is too rocky. Our chief object is to demonstrate how well this ore will concentrate, as we have very large reserves of ore too rocky for smelting, but capable of paying a handsome margin for concentrating.

Our laboratory is also situated here, although it will soon be moved to a much more commodious building at the smelter, where all our samples will be prepared by machinery, and where our electricity for assaying will be furnished by a small dynamo, thus avoiding the annoyance and uncertainty of batteries for that purpose. I need hardly mention that all our copper-assaying is done by electricity, but I think it is more unusual to assay nickel in the same manner. Our chemist, Mr. F. L. Sperry, has succeeded in perfecting a method for so doing.

The third class of deposits is represented by the Evans mine, situated about one mile southwest of the Copper Cliff. This mine contains a large body of pyrrhotite which is nearly as massive in places as the lower-grade Stobie pyrrhotite, while it runs high in nickel. The copper-contents is also very satisfactory in many parts of this deposit; and although it is by no means fairly developed yet, I cannot but regard it as an exceedingly valuable property. The company has valued this mine highly from the commencement, and has thought it worth while to provide it with one of the finest shaft- and rock-houses in the country. The ore will be hoisted in cars, on a platform-hoist, to the top of the high rock-house, where it is dumped 12 feet on to the breaker-floor, which will hold several hundred tons. The breaker-jaws are on a level with the floor, and it is fed with great ease. The ore being automatically sieved, as at the Copper Cliff mine, the different sizes fall direct into separate bins, whence it flows direct into the railroad cars by merely lowering the respective steel chutes. At present, we hoist with a kibble merely, but even this small affair gives us some 100 tons daily of

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ore. We have here a 3-drill compressor and two good pumps. I regret to say that both the Copper Cliff and this mine make considerable water.

Roasting.

The metallurgical treatment of this ore begins at the roast-yard, where the ore is roasted in heaps to remove the greater part of the sulphur, and oxidize the iron present as far as practicable. This roast-yard has been made at much trouble and expense, owing to the rough and rocky nature of the ground, and the fact that the only proper site for it was heavily wooded. Two railroad-tracks of standard gauge stand in connection with it, one being a high trestle-track which extends longitudinally over the entire roast-yard, the other being situated 4 feet lower than the roast-yard, and extending parallel with the upper track, along one edge of the yard. The upper track is to deliver the raw ore from the mines. As soon as we can make the necessary arrangements it will be brought in dump-cars and dumped on to elevated platforms along the side of the track, whence it will be wheeled to the heaps which extend in a long row at right angles to the tracks. The roast-yard is nearly half a mile long and 100 feet wide, so that the length of the piles is limited by the width of the ground. After allowing space to get around them and for drains, about 80 feet is left for the length. They are about 40 feet wide, and as the ore is piled about 7 feet high on the wood, will hold about 800 tons. They are built in the usual manner, about 30 cords of wood being sufficient to kindle a pile. After the main body of the pile is built up of coarse ore, a layer of ragging or medium ore is put on, 6 inches to a foot thick, according to the supply on hand, and this is covered in the usual manner with fines. By interposing a layer of rotten wood and chips between the ragging and fines, we are enabled to roast both of these smaller sizes more perfectly that is usually done. In general, we find the whole heap well enough oxidized to take it direct to the smelter without re-roasting any portion of it, which contributes materially to the economy of the operation.

A heap of 800 tons burns about sixty days if properly managed. Very great care has to be exercised in this operation, or the combustion becomes too rapid, and a great part of the sulphides in the ore melts down into a solid matte, which is most difficult to break up, and which carries far more sulphur than is permissible. Almost the entire success of the smelting-process depends upon a good roast.

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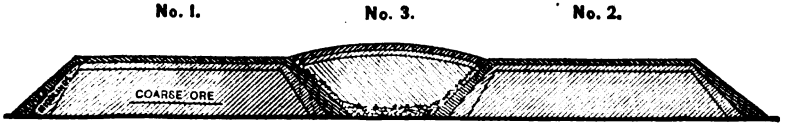
If the sulphur is not properly removed, a great quantity of low-grade matte is formed, into which the iron goes, leaving the silica without sufficient flux, and making the furnace run slowly and badly. If it is reduced to the normal amount of 7 or 8 per cent. of sulphur in the roasted ore, a rich matte is formed in comparatively small quantity, thus lessening freight and treatment charges. The iron which was combined with the sulphur is thoroughly oxidized, and is thus in a condition to combine at once with the silica, forming exactly the flux required, and making a rapid, clean and fluid run in the furnace. The importance of this process must be my excuse for devoting to it so much space, but thousands of dollars are lost every year in metallurgical operations for want of attention to this most difficult and delicate process.

The ore is roasted by contract at a very small figure, both for fuel and labor. By a second contract it is dug out of the heaps, which are frequently so fritted together as to require light blasts to loosen it up. The roasted ore is wheeled a few yards to the cars, which are standing on the lower-level track already described. Thence it is pushed by the engine up a rather heavy grade on to the track, which runs over the bins back of the smelter. The bins are calculated to hold nearly a week's supply.

The new method of heap-roasting, introduced at these works by the superintendent of smelting, James McArthur, may be easily understood from the accompanying sketch, in which Nos. 1 and 2



The V-Method of Heap-Roasting.

represent heaps built in the ordinary manner. They are allowed to burn out about one-half, and become thoroughly cooled on the sides. Then heap No. 3 is built in the passage-way between them, being shaped like a V. A bed of wood on the bottom, and a single layer of the same on the sloping sides of the two lateral heaps, provide ample fuel to start No. 3, which not only undergoes a thorough burning itself, but also sets the unroasted sloping sides of the two adjoining heaps on fire again, and thus roasts something like 50 to 100 tons of material that ordinarily is nearly raw. This method, besides greatly lessening the percentage of unroasted ore, also adds

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some 60 per cent. to the capacity of an ordinary roast-ground. For convenience in speaking of this new plan, we have called it "the V-method" of heap-roasting. Our best metallurgists are much pleased with the results obtained at Sudbury by this innovation.

Smelting.

I think the first and most striking point in connection with the furnaces of the Canadian Copper Company is the small size of the building covering them. There are two smelting-furnaces and two buildings; but as they are practically duplicates of each other, it will suffice to describe one. The entire building is 40 by 65 feet, of which 40 by 35 is on a lower level, and contains the furnace, while the rest of the floor is 8½ feet higher, and is devoted to the ore- and fuel-bins. When it is considered that the furnace averages 125 tons per twenty-four hours on favorable ores, I think that the space it takes up is unusually small, although amply large for all requirements. The furnace itself is a steel water-jacket of the Herreshoff patent, made by the Jenckes Manufacturing Company, of Sherbrooke, P. Q. It is rectangular, with rounded corners, and a slight convexity all around, so that it really approaches an oval. Its section, at the tuyeres, is 3 by 6 feet, and it has eleven 2½-inch tuyeres, there being five on each side and one at one end, the discharge-opening being at the other end. It is 6 feet high from tuyeres to charge-door, and is an unbroken water-jacket the entire distance from the cast bottom-plate to the charging-door. Above the threshold of this opening is a housing of boiler-iron, lined with fire-brick, which lasts as long as the furnace does. The charge-door is situated at one long side of the furnace, while the flue opening is opposite to it. The entire flue, as well as the iron charging-platform, rest on a series of girders and I-beams, which are supported by the stone walls of the building and by three iron columns. The latter are located so close against the furnace as to be entirely out of the way. The red-brick flue enters into a series of zigzag dust-chambers outside of the building. They are connected with a stack of the same material 60 feet high and 5 feet square inside from bottom to top. Its diminution in size, due to the taper of its walls, is fully compensated by the increasing thinness of its walls in height, they being only 8 inches thick for the top 20 feet. All the brick-work is so thoroughly ironed that, although in operation for nearly nine months, no crack of any moment has shown itself, and it promises to last indefinitely. This

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is due to extreme care in preparing the foundations in this frosty climate, and to proper ironing.

The water-space in this furnace is only 2 inches wide instead of 6 or 8, as is often the case, and we experience no trouble in consequence. The chief peculiarity of this furnace is its front connecting-reservoir or "well," as we term it. It is a circular, cast-iron, water-jacketted vessel, mounted on four stout wheels, and so designed that its hole in one side connects directly with the outlet-hole of the furnace. This forms a connecting channel, a few inches in length, thoroughly protected by water-cooling, through which the molten slag and matte flow out of the furnace as rapidly as they are formed. They thus escape the influence of the blast, and any possibility of the formation of the great bugbear of copper-smelters, great masses of metallic iron, called "sows" or "salamanders," is completely avoided. Smelting is thus robbed of half its terrors. Many of the new varieties of furnace have some similar provision, but none so convenient and perfect as this.

The slag and metal separate very perfectly in this quiet, spacious reservoir, and the slag flows in a continuous stream over the jacketted lip of the same at a height of some 10 inches above the outlet-hole of the furnace. This ingenious arrangement completely traps the blast, and, owing to it, we never have any foul slag to re-smelt from one week's end to the other. The matte is tapped at intervals of ten to twenty minutes through a separate bronze water-cooled tap-hole casting, which is bolted to one side of the well, and which is plugged with clay in the usual manner. Owing to its proximity to the hot stream of molten matter from the furnace, the tap-hole never chills. Instead of the ordinary sledging and labor every time the operation of tapping takes place, the furnace-man with us simply drives a ½-inch steel bar through the clay plug with a few light taps of a carpenter's hammer. The matte flows quietly into a slag-pot, and the small tap-hole in the casting is closed, without any chance of failure, by a clay plug as usual. There is no interruption of the blast, and none of the ordinary excitement and confusion so generally attendant on the operation of tapping. A slight matter, but one worth mentioning, is our arrangement for the better preservation of the cast-iron plates which form the floor of the building around the furnace, and which must be very smooth and solid, so as to easily draw heavy pots of molten material over them without spilling it. Usually, whenever a potful of slag is drawn away from the furnace, and before a new pot can be run in in place of the old one,

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a considerable amount of slag drops upon the iron plates. In spite of more or less protection from sand, they soon become so warped and out of shape that within three or four weeks they are totally destroyed, and have to be replaced by new ones at a considerable expense of time and money. To obviate this universal difficulty, we have cut a circle in the plate just where the stream of slag naturally falls, and have introduced into this a shallow cast-iron basin. The slag drips into this without injuring the plates, and is taken out, from time to time, by the fork and thrown into the slag-pot. The basin stands two months or more before it is destroyed, and is replaced at a cost of 75 cents, and without a moment's loss of time to the furnace.

There is nothing particularly worthy of notice about the charging-floor except what I have already described. Every pound of ore and fuel is accurately weighed on a 5-beam charging-scale, and is also sampled. In the same way the matte produced is daily sampled, as well as accurately weighed, before it is dumped from the pot, and the slag is sampled from every potful and assayed once every twenty-four hours. From these data we are able to prepare an accurate daily history of the furnace-work in each furnace for the benefit of the directors at home, to whom it is sent in a weekly sheet.

By properly mixing our three different ores, we are enabled to get along without ever using a pound of flux, which is a very fortunate circumstance, as I know of neither limestone or iron-ore in the vicinity suitable for this purpose.

For fuel, we use Pennsylvania coke of the best quality, which is brought *via* the Great Lakes and the Algoma branch of the Canadian Pacific Railroad at a less cost than might be imagined. We smelt 7 to 8 tons of ore to 1 ton of coke, as proved by having our coke overrun when we clean up, and estimate our large shipments of some thousands of tons, for there is very apt to be a heavy deficit in coke.

Our furnaces are supplied with wind by two separate No. 6 Baker blowers, one to each furnace. The blower to our No. 2 furnace is provided with a vertical engine, which forms a part of the blower itself, while our No. 1 furnace is run by a large Brown automatic engine, which has ample power for other additional work. Our water is pumped from a neighboring stream, where we have built a dam and formed a small lake, by two duplex Worthington pumps, either of which has ample capacity to supply both furnaces. Two

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steel tubular boilers, set in one battery, supply steam for the entire plant.

If I were asked to point out the most notable features of these works, apart from the mines, I should mention the very large capacity of the furnaces, nearly or quite 125 tons per twenty-four hours for each on fair ores; the entire absence of flux, owing to a more or less basic gangue-rock, and some very massive pyrrhotite from one of our mines, the Stobie; the very favorable ratio of ore to coke, 6 or 8 to 1, not as the result of an especial trial under favorable circumstances, but from months of average work. The same may be said of the capacity of the furnaces, which is greater than any that I know of running on pyritous ores.

Of course, the chief peculiarity of our product is the large percentage of nickel which our matte carries, in addition to its copper contents. Our average matte has about 27 per cent. of copper and 15 per cent. of nickel; and both of these metals are of such purity that, when separated, they are fitted for the most delicate work, and stand any tests that any similar metals stand. As every copper-smelter knows, we can determine the percentage of copper matte by eye with such exactness, that, from any per cent. between 25 and 80, I have never cared for an assay of it for my own benefit, as my eye will always tell me its copper-contents near enough for metallurgical or chemical purposes. But even a very slight admixtnre of nickel completely masks the color and texture produced by varying percentages of copper; and in our mattes I do not think that any one could distinguish between a matte carrying 15 per cent. and one carrying 40 per cent. of copper, providing that nickel were also present in our usual proportions.

At present, we break our matte into large pieces, which we pack into old oil-barrels, chinking up the interstices with the smaller fragments. These are shipped to Swansea, or to various German works, according to the bids of the foreign refiners for our product.

It may perhaps be newe to some of my hearers that the entire world's production of nickel annually is less than 1000 tons, the bulk of this being produced by the New Caledonian nickel-mines, which are oxide-deposits situated in a serpentine dyke. While I am not at liberty to give figures of our production, it will still be very evident to any observer that the world's production of this year will be a good deal larger than that of 1888; and as we are told that New Caledonia is losing money at the present low price of the metal, it is evident that we need not look far for the world's supply.

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*NATURAL GAS EXPLORATIONS IN THE EASTERN ONTARIO PENINSULA.**

BY CHARLES ALBERT ASHBURNER, PITTSBURGH, PA.

(Ottawa Meeting, October, 1889.)

Gas in the St. Lawrence Valley.

Natural gas has been known to exist for many years in the St. Lawrence Valley, between Quebec and Montreal, and more particularly in the vicinity of Louisville and Three Rivers, 74 and 94 miles respectively northeast of Montreal, and 98 and 78 miles respectively southwest of Quebec; the distance from Montreal to Quebec along the St. Lawrence River being about 175 miles.

It was not until 1880 that any practical explorations were begun with the hope of finding gas in sufficient quantities for commercial uses. In this year Messrs. Piret and Genest, of Three Rivers, sank a well near St. Maurice to a depth of 50 feet, where solid rock was encountered. A similar well was drilled through the glacial and alluvial drift, in the same district, in 1883, by the Messrs. Renaud Frères and Dubois, to a depth of 70 feet before solid rock was encountered. In this latter well a strong flow of gas was obtained. The gas was an outflow from a very limited reservoir in the porous glacial gravel, which had been filled from the gas resulting from the decomposition of vegetable remains, which had been buried in the gravel at the time of its deposition. Subsequently, other wells were sunk in the same way, with the hope of getting gas in commercial quantities from similar reservoirs.

Commercial Gas.—By gas in commercial quantities I mean gas in such quantity that, if utilized with proper care and economy, it would generate steam at a cost below the cost at which the same duty could be obtained from the use of coal or wood at current prices, and that

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* This paper has more special reference to explorations for natural gas in that portion of Ontario bounded on the north by Lake Ontario, on the east by the Niagara river, on the south by Lake Erie, and on the west by a line drawn from Hamilton, southwest to Port Dover.

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the net revenue from the use of the gas would be such as to give a reasonable profit on the capital invested in the drilling of the well and the piping of the gas to the consumer, and at the same time provide for a sinking fund to replace this invested capital within the time when it might be considered that the gas-producing rock would become absolutely exhausted. It is important in this connection to bear in mind that natural gas, like all other mineral deposits, can be exhausted. In a commercial sense a gas-reservoir is not unlike a coal-bed. For every cubic foot of gas taken out of a gas-reservoir there is just one cubic foot less remaining.

Relation Between Petroluem and Natural Gas.

A popular impression exists that the gas is being continually produced, but this is not a fact in a commercial sense, as far as the natural gas-deposits of the Appalachian region are concerned. Natural gas and petroleum are intimately associated with one another. In all gas-reservoirs it is possible to find petroleum, sometimes, however, in such infinitesimal quantities that it is of no commercial value, and can only be found by a very careful examination of the gas-producing rock. Natural gas is also always to be found in the rocks which produce petroleum, although in the latter case the amount is so small as not to be practically valuable.

While it is unnecessary in this connection to comment upon the question whether the primary condition in which these hydrocarbon products exist is gas or oil, and if gas, whether the secondary product is petroleum, and if petroleum, whether the secondary product is gas; still it is interesting to know that wherever petroleum has been found a certain amount of gas is always being produced by the volatilization of the lighter products of the petroleum. This production of gas, however, is so small that the statement is justified that no gas is being produced in quantities sufficiently great to be of commercial value. In disproof of the opinion, which I have always held, that petroleum can be found in gas-producing rocks, the statement is frequently made by practical gas men that in certain districts where natural gas has been found, and where it is being practically utilized for commercial purposes, the gas is absolutely dry; that is, it contains no petroleum which can be obtained as a precipitate from the gas. While it is true that in many of our prominent gas districts the petroleum is not readily visible in the gas, still I know of no case where I have not been able to find some

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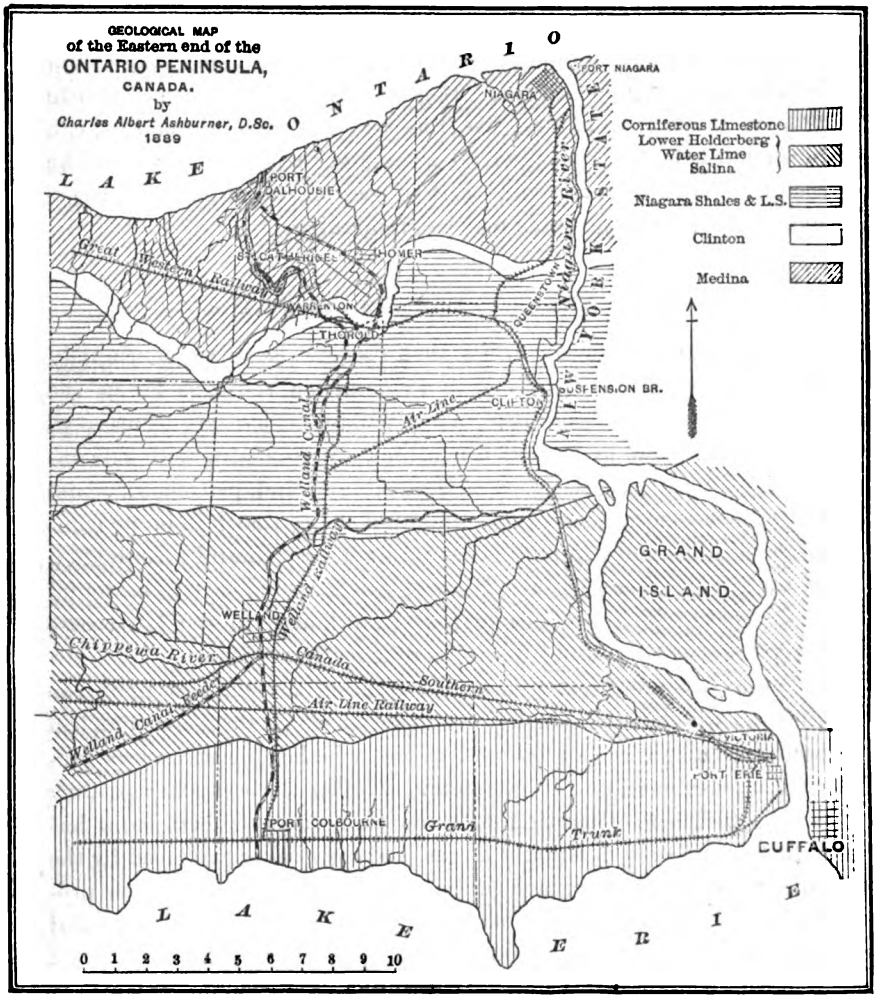
petroleum, even if it were only a single drop, in a district which is producing daily millions of cubic feet of natural gas.*



Without making further reference to the occurrence of petroleum in the district under consideration, I believe that whatever may be

* I have given careful thought to this question in connection with the occurrence of the immense deposits of natural gas in the vicinity of Pittsburgh, Pa., and the great possibility which, to my mind, exists of finding valuable pools of petroleum near Pittsburgh, as well as new pools of gas. It should be a sufficient warrant for a very exhaustive geological examination of the district, at the expense, however great, of the Pittsburgh natural-gas companies.

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the possibility of finding natural gas in commercial quantities in the district, no petroleum will ever be discovered that can be economically utilized.

Gas Between Montreal and Quebec.—The gas found in the vicinity of Louisville and Three Rivers, in the shallow drift wells, was at first thought to be similar in its occurrence to the rock gas found in Pennsylvania. The first intelligent reference to the occurrence of this gas was made by Rev. Mr. La Flamme, Assistant Geologist of the Canadian Geological Survey, and Professor at La Valle University, in 1883, when he read a paper at Louisville by the light of the gas obtained in the vicinity. He called attention to some of its geological relations. It was soon realized that these drift-gas reservoirs would not supply sufficient gas for practical purposes, and attention was directed to the possibility of finding a larger quantity in the underlying Trenton limestone, which forms the middle member of the Siluro-Cambrian period. It might be well to state that nowhere in America has drift-gas been found in quantities which could be profitably drilled for and utilized, except in some sections of Illinois; and even there the quantity of gas is so limited that it can only be profitably utilized for domestic illumination and heating in the immediate vicinity of the wells.

A number of natural-gas companies have been organized in the United States to develop the drift-gas pools, and to distribute the gas for general consumption in small towns in the immediate vicinity of the wells. In each case, however, to which my attention has been directed, the investment has proved very unprofitable, and the experience had in the United States should discourage any one from attempting to utilize the drift-gas in Canada. In the latter country the quantity which has so far been found is much more limited than in the United States, where the money which has been invested for drilling the wells and piping the gas to any distance has been absolutely thrown away.

All the gas which has ever been found, and which can be profitably utilized in the St. Lawrence Valley southwest of Quebec, comes from the Trenton limestone strata, or immediately overlying Utica slates. The recent discovery of large quantities of gas in the Trenton limestone, particularly in the States of Ohio and Indiana, has increased the search for gas in the same geological horizon in the St. Lawrence Valley.

The discovery of the existence of gas and petroleum in the Trenton limestone is generally accredited to the parties who first drilled

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oil- and gas-wells to the Trenton limestone in Ohio and Indiana, about eight years since. The existence of both oil and gas in the Trenton limestone has been known for many years; both Prof. J. P. Lesley, State Geologist of Pennsylvania, and Dr. J. S. Newberry, late State Geologist of Ohio, pointed out more than twenty-five years ago the occurrence, in the State of Kentucky, of both oil and gas in the Trenton limestone strata. At that early date they suggested the possibility that petroleum could be found in some sections of Kentucky in profitable quantities.

The first extensive development of petroleum and natural gas in the Trenton limestone, however, occurred in Ohio; and this led to the drilling of wells in the St. Lawrence Valley in the vicinity of Louisville and Montreal to depths varying from 550 to 1500 feet. In them gas was found at varying depths, but these early operations were badly conducted. The wells were not carefully cased, water was permitted to flow into the holes and partially drown out the gas-flow, and little was proven except that gas did occur in the strata drilled through.

From the facts which I collected in an examination of the St. Lawrence valley between Montreal and Quebec I was led to conclude, that while there was every hope of getting gas in special localities, which could be profitably utilized for light and heat by domestic and manufacturing consumers in the immediate vicinity of the wells, there was no promise that it would ever be found in quantities large enough to permit of its profitable transportation in pipes to any distance from the wells, and that certainly gas would never be found in the immediate vicinity of either the city of Quebec or Montreal, or near enough to these cities to allow it to be piped profitably to either city for municipal distribution for heating or even illumination.

The fact that the Trenton limestone had proved such a productive oil- and gas-horizon in the States of Ohio and Indiana, and that gas had been found in limited quantities between Quebec and Montreal in very nearly the same geological horizon, was one of the suggestions leading to a search for gas in other localities in Canada where the geological conditions might be more favorable than in the St. Lawrence Valley. I wish to make a general reference to other suggestions which led to the same inquiry.

Petroleum and Gas in Western New York.

In Allegany County in western New York as early as 1865, an organized effort was made to find the oil-sands which had proved so

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productive in the Oil Creek district of Pennsylvania, and a well was drilled in the vicinity of Independence. It proved to be a complete failure as far as it concerned the discovery of oil in profitable quantities. In 1874 extensive petroleum pools were discovered in McKean County in Pennsylvania, directly south of the Pennsylvania-New York State line, and about 70 miles a little east of south of Buffalo. This led to renewed efforts to discover oil in New York. They were successful as far as Allegany County was concerned, in which a pool was found which has been very productive and profitable; but the wells drilled in all other localities in western New York were failures in respect to the developing of profitable deposits of petroleum.

In 1880, when natural gas began to be generally used as a fuel, the search for oil and gas in New York received an immense impetus and many wells were put down. A few favorable localities were found for the occurrence of natural gas in commercial quantities between Albany on the Hudson River, and Buffalo at the eastern end of Lake Erie. Facts relating to these explorations have already been communicated to the Institute in a paper* which I read over two years ago at the Duluth meeting. In 1883 wells were drilled in the city of Buffalo, and natural gas was found at depths ranging from 415 to 460 feet in the lower part of the Salina lime shales. These wells are particularly referred to in a paper read a year ago at the Buffalo meeting on "The Geology of Buffalo as Related to Natural Gas Explorations Along the Niagara River."†

Explorations West of the Niagara River.

These facts pointed to the possibility of finding gas in commercial quantities on the eastern end of the Ontario peninsula between Lakes Ontario and Erie. Wells were put down at Port Colborne at the southern end of the Welland Canal with the hope that gas would be found at the same geological horizon as at Buffalo, and if not, that the Trenton limestone might be encountered by drilling to depths not exceeding 1500 feet. I referred to this in the paper which I read at the Buffalo meeting.

The difficulty which the local geologists experienced in harmonizing the section of one of Port Colborne's wells which was drilled to a depth of 1500 feet, with a section of the Buffalo wells, and the conflicting opinions held by geologists at Buffalo as to whether the

* *Trans.*, xvi., 906.

† *Trans.*, xvii., 398.

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deepest well in Buffalo (1305 feet) stopped in the red rocks supposed to exist in the lower part of the Salina group, or in the red Medina sandstone, together with conflicting views held as to the geological relationship of the gas found at Port Colborne at a depth of 760 feet, and at Buffalo at a depth of 460 feet, and the failure to encounter the Trenton limestone in the 1500-foot well at Port Colborne, produced the greatest confusion as to the understanding of the geological structure along the Niagara River and in the Ontario peninsula east of a line drawn from the city of Hamilton at the extreme western end of Lake Ontario southwest to Port Dover on the northern shore of Lake Erie and west of the city of Buffalo.

It was held by many intelligent persons, that the gas found at Buffalo and Port Colborne was really Trenton gas, and that it had leaked from the original Trenton limestone reservoirs into the higher strata in which it was found, and since the general upward dip of the strata along the Welland Canal is toward the north, it was decided by local geologists and capitalists to drill a well at Thorold in order to explore the Trenton limestone, which it was correctly concluded had not been found in the Port Colborne 1500-foot well, and which must be found at Thorold at less depth than at Port Colborne.

After the Thorold well had been drilled to a considerable depth, I was consulted by citizens of St. Catharines as to the possibility of obtaining gas in the immediate vicinity of that town in sufficient quantities to permit of its profitable distribution through the city for manufacturing and domestic consumption.

The interest which the citizens of St. Catharines took was local, and presented itself as follows:

First. Was it possible that the gas which had been found in the region surrounding St. Catharines could be found in the strata underlying the town, or sufficiently near to make it profitable to pipe the gas to the town?

Second. What promise was there of finding gas under St. Catharines in the same geological horizon in which it occurs in western Pennsylvania and in western Ohio?

Third. Where, in the vicinity of St. Catharines, was the best place to drill a test-well in order to find the gas if it existed, or to conclusively settle the question that it did not exist without making further explorations after one well had been drilled?

General Geology of the St. Catharines District.—The general geological structure of the Ontario Peninsula, west of the Niagara River,

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is comparatively simple. The strata along the Welland Canal which outcrop, or which immediately underlie the Saugeen and Erie clays, are composed of the representatives of the Lower Devonian and Upper Silurian formations, commencing at Port Colborne with the Upper Helderberg or Corniferous limestones, and descending the geological scale toward Port Dalhousie, where the Medina sandstone is found. The individual groups are as follows:

Upper Helderberg limestone,	No. VIII.
Oriskany sandstone,	(Wanting) No. VII.
Lower Helderberg limestone,	(Wanting)
Salina or Onondaga shales,	No. VI.
Niagara shales and limestones,	
Clinton red and gray shales,	No. V.
Medina sandstone,	No. IV.

The geological structure of these groups is such that they overlap each other successively in going from Lake Erie north to Lake Ontario, having a general dip toward the south on an average of 35 feet to the mile. The strata extending from the Niagara limestone down into the Medina sandstone for a vertical distance of 350 feet are exposed along the Niagara gorge from the Rapids above the Falls to the mouth of the Niagara at Fort Niagara. The general dip of these strata to the south is modified apparently by two systems of local rolls, one having a general northeast and southwest direction, and the other system, which is very much less prominent, having a northwest and southeast direction. These local rolls, which are hardly to be dignified by the names of anticlinals and synclinals, change the direction of the dip and its intensity at local points.

If gas shall be found at any point, these two systems of rolls will have an important bearing upon the location of the most productive wells.

No wells have ever been drilled in the immediate vicinity of St. Catharines to explore especially for either oil or natural gas, but three wells have been drilled at different times to explore the strata generally for any minerals of value which they might contain, and particularly salt.

About 1818 the drilling of a well was commenced in the town on a property which now belongs to Mr. J. P. Merritt. Drilling was continued at this well at various times until 1862. The total depth drilled to was 300 feet, and I was informed by Mr. Merritt that the total cost for drilling was $4000. Bed-rock was struck in this

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well at a depth of about 15 feet, and numerous mineral springs were encountered throughout the whole depth of the well.

About 1855 a well was drilled on the property which now belongs to the Stephenson House to a depth of 510 feet, with the hope of obtaining brine for the manufacture of salt. The diameter of this hole was 5 inches, and it is reported to have traversed the red strata of the Medina formation to a depth of 450 feet when shales were struck which were supposed to represent the Hudson River formation, but I am disposed to believe that the Medina strata continued to the bottom of the well. In 1862 the well is reported to have yielded 22 gallons of water per minute. The brine is so much charged with lime and magnesia salts as to be unfit for the manufacture of salt, but the water has acquired a great reputation in the treatment of diseases, being taken internally and used for bathing. The specific gravity of this water is reported to be 1.036.

A third well was drilled in St. Catharines in 1861 by Mr. E. S. Adams, in which was discovered a saline water somewhat similar to that obtained from the Stephenson well, but somewhat weaker. The table below gives the analyses of the water from these two wells.

Analysis No. I. was made in 1860 by Prof. Croft, of Toronto, of a specimen of the Stephenson well water, while specimens Nos. II. and III. were made in December, 1862; No. II. being a specimen of the water from the Stephenson well, and No. III. being a specimen of the water from the Adams well. These analyses gave grains of mineral constituents for 1000 parts of each water as follows:

	I. Grains.	II. Grains.	III. Grains.
Chloride of sodium,	29.8034	23.00	19.94
" potassium,	.3555	trace	trace
" calcium,	14.8544	9.66	6.49
" magnesium,	3.3977	2.40	1.95
Iodide of sodium,	.0042	trace	trace
Sulphate of lime,	2.1923	1.75	1.77
Total,	50.6075	36.81	30.15

Analyses Nos. I. and II. show a considerable variation in the composition of the water from the Stephenson well within a year. It is more than probable that at the time the second specimen was collected some surface fresh water had leaked into the well.

The question naturally suggests itself, which of the strata immediately underlying the region around St. Catharines occur elsewhere

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and contain gas in commercial quantities. The Clinton sandstones and shales are exposed in the Government quarry at Merritton, back of the Riordan paper mills. Immediately overlying the Clinton sandstone is an exposure of the Niagara limestones and shales. The elevation above tide of the geological plane dividing the Clinton from the Niagara formation in this quarry is 490 feet. This same plane, under the town of Port Colborne, was passed through in the Port Colborne well No. 3 at a point 125 feet below tide, so that there is a total dip in the strata between Merritton and Port Colborne of 615 feet in a downward direction toward Port Colborne. The average dip per mile between these two towns is, therefore, 35 feet. It must be remembered, however, that there are some local rolls in the strata between these two towns, which have already been referred to, which make the dip less than the average rate of 35 feet per mile in some localities, and in other localities more than the average rate of 35 feet per mile.

Judging from exposures in the vicinity of St. Catharines, and as far north as the south shore of Lake Ontario, this average rate of dip continues north of the town and at least as far as the south shore of the lake. These dips, in themselves, are favorable to the existence of natural gas, and, judging from a consideration of the geological structure of the rocks themselves, it is safe to conclude that the strata which underlie the St. Catharines region would be found sufficiently free from cracks and sufficiently impermeable to the upward flow of petroleum or natural gas to make 500 feet of strata, ample to confine any oil or gas in a possible reservoir-rock which they might overlie.

The only strata underlying the St. Catharines region which, in Ohio, New York and Pennsylvania, contain gas in commercial quantity are the lower part of the Medina sandstone, the Hudson River shales, slates and sandstones, the Utica shales and slates, and the Trenton limestone.

The outcrop of the top of Medina sandstone occurs south of the town. Therefore it was evident that the gas horizon found at Buffalo from 15 to 60 feet above the bottom of the Salina lime shales; at Port Colborne, in the top of the Medina sandstone; at Getzville, 6 miles from North Tonawanda, in the Niagara lime shales; and, at Fredonia, New York, in the Corniferous limestone, could not be found at St. Catharines, since none of these formations underlie the town. As for the probability of the existence of gas in the Trenton limestone, nothing could be said other than that the

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geological structure of the district under examination was favorable to its existence; the fact must be determined by actual drilling.

Geological Section of the St. Catharines Well.—After one well was drilled, an examination of the mineralogical and chemical structure of the specimens of the rock obtained from it would enable deductions to be drawn for a considerable area. The geological structure about 1½ miles west of the town was found to be more favorable than elsewhere, and a well was drilled at a selected point.

Immediately subsequent to the erection of the Thorold well derrick, and before drilling had been commenced, I ventured an opinion that the geological structure of the strata at Thorold absolutely precluded the existence of gas there in commercial quantity, and that the drilling of this well would not settle the question conclusively, because even if a gas pool did exist in the Trenton strata under the eastern portion of the Ontario peninsula, there was no hope of encountering it in the Thorold well.

The St. Catharines well was located at a point where there was apparently a very low, but perceptible, dome in the strata, caused by the intersection of two broad and comparatively flat anticlinals, one having a general north-northeast and south-southwest direction, and the other having a general northwest and southeast direction. After setting the drive-pipe in this well in the alluvium clay, drilling was commenced in the upper portion of the Medina sandstone strata, 130 feet vertically below the top of the Medina formation. The upper part of the Medina is boldly exposed at several points southeast of St. Catharines.

The St. Catharines well was drilled to a total depth of 2200 feet. The geological horizon separating the Medina sandstone from the underlying Hudson River shales was pierced at a depth of 720 feet. The Trenton limestone was struck at a depth of 1505 feet. The bottom horizon of the Trenton* was passed through at a depth of 2182 feet, drilling being stopped at a depth of 2200 feet after 18 feet of the Calciferous sandstone had been pierced.

Combining the sections of the Port Colborne and Buffalo wells with that of the St. Catharines well, the following is a general section of the strata:

* No attempt has been made to subdivide the limestone strata between the Utica shales and the Calciferous sandstone into the Trenton, Black River, Birdseye, Chazy and Quebec; the drillings between the Utica and Calciferous being so nearly alike, the entire 677 feet of limestones have been grouped for convenience under the name of Trenton.

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	Devonian.	Thickness. Depth.
No. VIII.	Corniferous limestone, ...	25 feet to 25 feet.
No VII.	Oriskany sandstone, ...	Wanting.
	Silurian.	
No. VI.	Lower Helderberg limestone, ...	Wanting.
	Water lime, ...	Wanting.
	(The Buffalo cement-bed is one of the top strata of the Salina, and does not belong to the water-lime group, as popularly supposed.)	
	Salina or Onondaga limestone and shales, .	500 feet to 525 feet.
	Niagara limestone, ...	155 " 680 "
	Niagara shales, ...	70 " 750 "
No. V.	Clinton limestone and shales, ...	50 " 800 "
No. IV.	Medina sandstone and shales, ...	850 " 1650 "
	Oneida sandstone and conglomerate, . . .	Wanting.
	Silurio-Cambrian and Cambrian.	
No. III.	Cincinnati, . . { Hudson River shales, Utica shales and slates. }	785 feet to 2435 feet.
No. II.	Trenton limestone, Black River limestone, Birdseye limestone, Chazy limestone, Quebec limestone, ...	677 feet to 3112 feet.
	Calciferous sandstone, ...	18 feet to 3130 feet.

The upper 930 feet of this section has been compiled from facts obtained from the Buffalo, Getzville and Port Colborne wells, and from a study of the strata along the Niagara River gorge from Lake Erie to Lake Ontario.

It may be of interest to note here that Corniferous limestone outcrops may be found in several of the higher hills in Buffalo and at Black Rock village (on the northern edge of the city of Buffalo), which derives its name from the character of the limestone at that point. The outcrop here occurs 52 feet above the surface of Lake Erie, or 625 feet above tide.

The Salina limestone and shales, upon which the Corniferous limestone immediately rests everywhere, in the vicinity of the Niagara River, underlies all the level country from Black Rock to a point about 2 miles south of Niagara Falls, covering a distance of 15 miles by the river. A stratum of the Niagara limestone, 75 feet below the top of the formation, forms the knuckle of the Falls. Between the Falls and the mouth of the river at Lake Ontario there is a vertical thickness of 550 feet of the Silurian strata boldly exposed, extending 350 feet vertically down into the Medina formation.

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The lower 2200 feet of the above section were classified from over 200 specimens of the drillings of the St. Catharines well and from a number of general facts reported of the strata pierced by the Thorold well.

In the St. Catharines well several small pockets of gas were found at a number of places in the Medina, Hudson, Utica and Trenton strata, and about 500 feet below the top of the Trenton limestone, at a point more than 1500 feet below ocean level, a pool of gas was pierced which produced on an average about 100,000 cubic feet of gas per day for several days, but was soon exhausted.

A careful mineralogical and chemical examination of the drillings from the Trenton limestone pierced by the St. Catharines well does not give promise of the existence of gas in commercial quantities in the Trenton limestone anywhere under the eastern portion of the Ontario peninsula unless the character of the rock changes very materially from that found in the St. Catharines well.

This section of the eastern portion of the Ontario peninsula, which is now published for the first time, is not only of great importance as bearing upon the occurrence of natural gas in the district, but one of the most important sections relating to American geology, since it conclusively settles the stratigraphical structure of the Niagara region, whose geology has been studied by more geologists and for a longer period than that of any district on the Continent. On account of the low southern dip of the strata (an average of 35 feet per mile), the lack of good exposures, except from the Falls north to the mouth of the Niagara River, and the fact that Lake Ontario cuts off the opportunity of observing the basset edges of the strata north of the river, it was impossible to determine absolutely the thickness of most of the strata from the Corniferous limestone down to the Trenton limestone except by drilling of wells; the records of the Buffalo, Getzville, Port Colborne, Thorold and St. Catharines wells, studied independently and conjointly, have afforded for the first time this favorable opportunity.

Upon the facts contained here and in my Duluth and Buffalo papers already referred to, the following conclusions bearing upon the occurrence of natural gas in the eastern Ontario peninsula are based:

Conclusions.

1. The gas in the Buffalo wells, in quantities sufficiently great to be profitably utilized, comes from the lime shales occurring from 15

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to 60 feet above the bottom of the Salina formation. Small amounts of gas, but not in commercial quantities, were also found in the Niagara limestones from 45 to 80 feet below the top of the Niagara formation.

2. The top of the Trenton limestone under the city of Buffalo should be encountered by a well drilled in the vicinity of the Buffalo Cement Company's present wells, at a depth of 2400 feet, more or less.

3. Little hope can be entertained of finding gas in commercial quantities in the Trenton limestone under the eastern Ontario peninsula.

4. In the southern portion of the Ontario peninsula, east of a line from Hamilton to Port Dover, natural gas, in quantities sufficiently great to be profitably utilized near the wells, can be expected to be found in special localities where the local geological structure is favorable to its existence, in the following geological horizons: 1. From 15 to 75 feet above the bottom of the Salina formation. 2. In parts of the Niagara limestone and shales; notably in the upper part of the Niagara limestone. 3. In the upper part of the Medina sandstone. 4. In the Hudson River and Utica shales and sandstones. The most promising horizons are those in the upper part of the Medina and lower part of the Salina formations.

5. It cannot be expected that gas would exist in any of the strata in commercial quantity unless the gas-reservoir stratum is covered by at least 400 feet of superincumbent strata; nor can it be expected that gas will be found in any two or three different strata in the same well.

THE DAVIS-COLBY ORE-ROASTER.

BY STERLING G. VALENTINE, PH.D., LEBANON, PA.

(Ottawa Meeting, October, 1889.)

THE annual use of large quantities of sulphurous iron-ores (over 1,500,000 tons in the United States) has led to a search for the best methods of desulphurization, in order to make this material more available for the manufacture of a good quality of pig-iron. Although such ores can be used raw in small amounts, they must be previously roasted if they are to constitute the entire ore-charge of a furnace.

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In a former paper * some of the chief conditions necessary to satisfactory roasting of ore were examined on the basis of experimental data. They indicated, too, the most prominent characteristics of a good kiln. It is proposed to give in this paper some description of the latest application of a gas-fired roaster to the preparation of sulphurous ores, as exhibited in the Davis-Colby kiln, including a statement of some results already obtained in actual practice. The principles involved in the construction and work of this roaster cover the most prominent points heretofore shown to be necessary for good results.

The Davis-Colby ore-roaster is the outcome of some years of trial and practical experience. In the years 1878–9 the Katahdin Iron Works built at their furnace a Westman kiln, which was fired with wood, for the purpose of making the ore used there available for the manufacture of charcoal car-wheel iron. The ore at that locality is a hematite, resulting from the oxidation of a ledge or vein of pyrites, and carries a good deal of sulphur. The Westman kiln being in many points unsatisfactory, changes made from time to time to secure increased efficiency resulted in what is now known as the Davis-Colby gas-fired roaster. This has been and is yet in constant use at the Katahdin Iron Works, and kilns of the same type have been since introduced at other points.

The illustrations, reduced from drawings kindly furnished by Mr. A. G. Davis, will give a clear idea of the construction of the kiln. In general it consists of two concentric shafts of brick-work, enclosing between them an annular space, *E*, 18 to 24 inches in section, to contain the ore under treatment. The inner shaft, *F*, is continued to a proper height above the top of the roaster to form the draft-stack, as shown in Fig. 2; or it may be covered, as in Fig. 1, the products of combustion being carried downward and out through the flue, *F'*, to a separate chimney. This would allow of any utilization of the fumes deemed expedient. A cone-covered top, as shown, permits of more convenient charging, as ore can be dropped from car-hoppers upon the cap, *L*, giving an even distribution of the ore. In the outer wall are placed the chutes, *C;* the fire-arches, *B;* the gas-flues, *X*; and the poking-holes and air-flues, *Z*. Openings in the inner wall, *D*, admit the fumes and products of combustion to the draft-stack, *F*, or to the flue, *F'*.[1] These openings are placed higher or lower, according to necessities arising from the

* *Trans.*, xviii., 78.

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character of the ore under treatment. The external shape, as shown in Figs. 2 and 3, is that of the Gjers kilns, which were re-

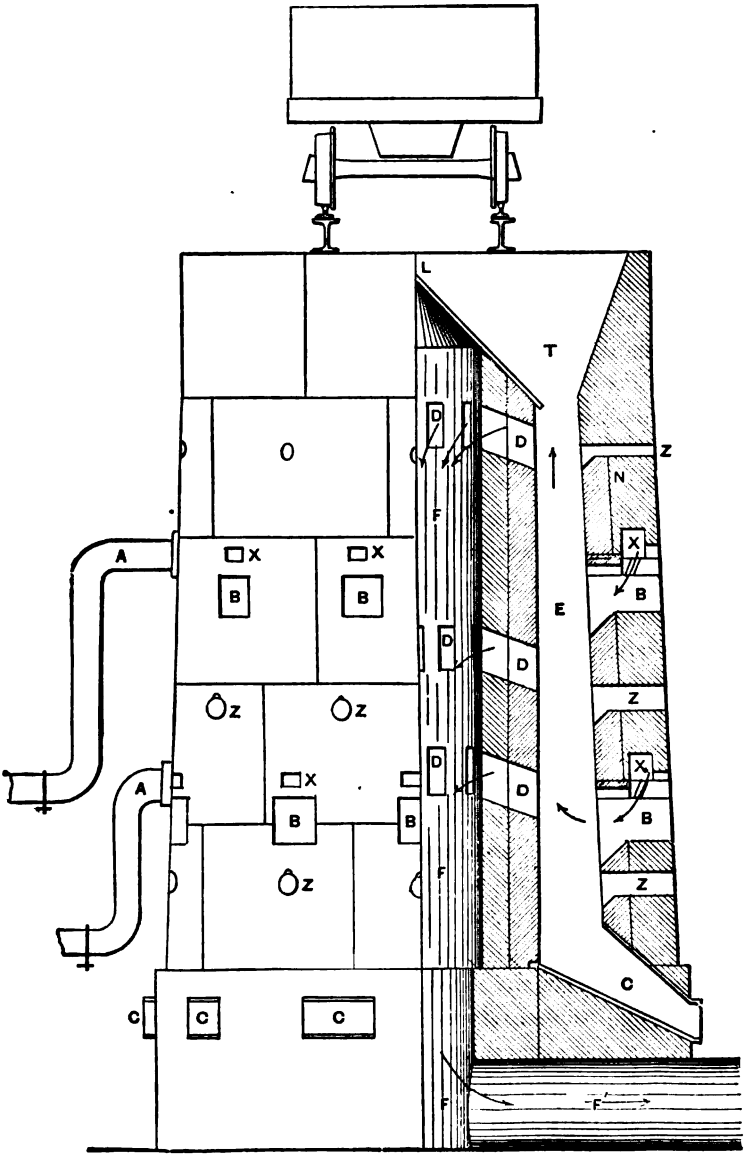

FIG. 1.—Down-Draft Davis-Colby Ore-Roaster.
(Scale: $\frac{1}{16}$ inch to 1 foot.)

modelled to the new type. These illustrations are given to show

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how readily the ordinary style of roaster may be adapted to this new design. In some cases, particularly where dense hard magnetites are to be roasted, the height of the kiln is increased above that shown, for the purpose of giving the ore a longer exposure to heat; and the fire-arches are put at a greater vertical distance apart, so that the ore may partially cool after its first heating, and be cracked or fissured, thereby exposing the remaining sulphur more thoroughly to the action at the second fire-arches. A gallery around the outside, at the proper height, gives workmen convenient access to fire-arches and gas-flues.

Gas from the mains, *A*, enters the flues, *X*, in the outer wall, and is distributed to each of the fire-arches, *B*, flowing downward through the outlets, as indicated by the arrows. Two or more sets of fire-arches are used. In these the gas is burnt, the flame and heat impinging on the ore as it descends in the annular space, *E*, raising it to any temperature required. This heating is done in the presence of abundant air, entering at the chutes, *C*, the fire-doors, *B*, and the poking-holes, *Z*. After passing through the ore the air enters the draft-stack through the openings, *D*, carrying with it the gaseous products of combustion and sulphur oxides from the ore.

The ore is filled in at the top, *T*, passes down the space, *E*, which gradually widens downward, and is drawn below from the chutes. Its cross-section being so narrow, the whole body of ore is thoroughly exposed to the fire.

At most places where these roasters have been introduced, the gas used for firing has been surplus gas from the blast-furnace. Where this supply is abundant and clean, or where the gas can be cleaned before introduction to the flues of the roaster, it answers very well, and is economical. Small blast-furnaces can the better allow this method, as their surplus supply of gas is generally sufficient, and likely to contain a larger amount of combustible constituents than that from large furnaces, the latter, as a rule, having little surplus gas above that required for hot-ovens and boilers. But there are serious objections to depending on the use of furnace-gas under the circumstances generally prevailing. The gas-flues and small openings into the fire-arches require to be kept open and free from dirt; and while this can, of course, be readily done, it increases the attention required in working the kiln, and is so far an annoyance and disadvantage. The difficulty of getting a clean furnace-gas is an important objection to its use. Then, too, the variations in the quantity and quality of furnace-gas are against it. When a furnace-

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man is specially desirous of securing well-roasted ore, he may find such fuel unequal to the task of preparing it. There are, after all, few furnaces that afford gas enough to heat their blast, furnish their steam, and roast their ore at the same time. It is questionable economy to make any endeavor to turn a blast-furnace into a gas-producer. It may be desirable at times, however, to use waste furnace-gas for whatever part of the work of roasting it may be equal to, and augment the supply from another source.

Producer-gas, therefore, is a preferable fuel for the roaster, and can be obtained cheaply from some such apparatus as the Taylor producer. With such an arrangement the gas-supply can be kept uniform in quantity and quality, or can be varied, as the case may demand, thus giving increased control of working.

The principles involved in the construction and working of the Davis-Colby roaster are well calculated to produce good results. The heat is under almost absolute control. Any set of fire-arches, or any single fire-arch, can have its supply of gas increased or diminished at will. The double set of fire arches serves to keep the ore at the desired temperature until drawn into the chutes, so that from the first point of heat it is held, during its entire descent, at such a temperature as to make the action of atmospheric oxygen upon it most effective.

The air required for a proper oxidation of sulphides in roasting is furnished abundantly through the poking-holes, fire-doors, and chutes, passing through the hot ore into the chimney. Thus the hot ore is always exposed to an oxidizing atmosphere, and a prime requisite for good results is well attained. The narrowness of the ore-section has the advantage of making it very difficult for any ore to pass the two or more fire-arches without being well exposed to oxidation. Since the openings into the draught chimney are just opposite and a little above the fire-arches, the heat and air are drawn through the whole of the annular space, all gaseous products of combustion passing promptly into the stack and being removed.

Sulphide of iron can be decomposed by atmospheric oxygen alone at ordinary temperature,—a very slow reaction, indeed,—but, like most chemical reactions, the process of oxidation is accelerated by heat. On the other hand, a high temperature, without oxygen, effects a partial decomposition of FeS_2, leaving a compound very nearly FeS, volatilizing about half the sulphur, but cannot carry the process further alone.

Oxygen being present, however, the chemical affinity between it

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and sulphur is increased by heat, and a complete oxidation of all the sulphur is theoretically attainable. Oxygen is absolutely neces-

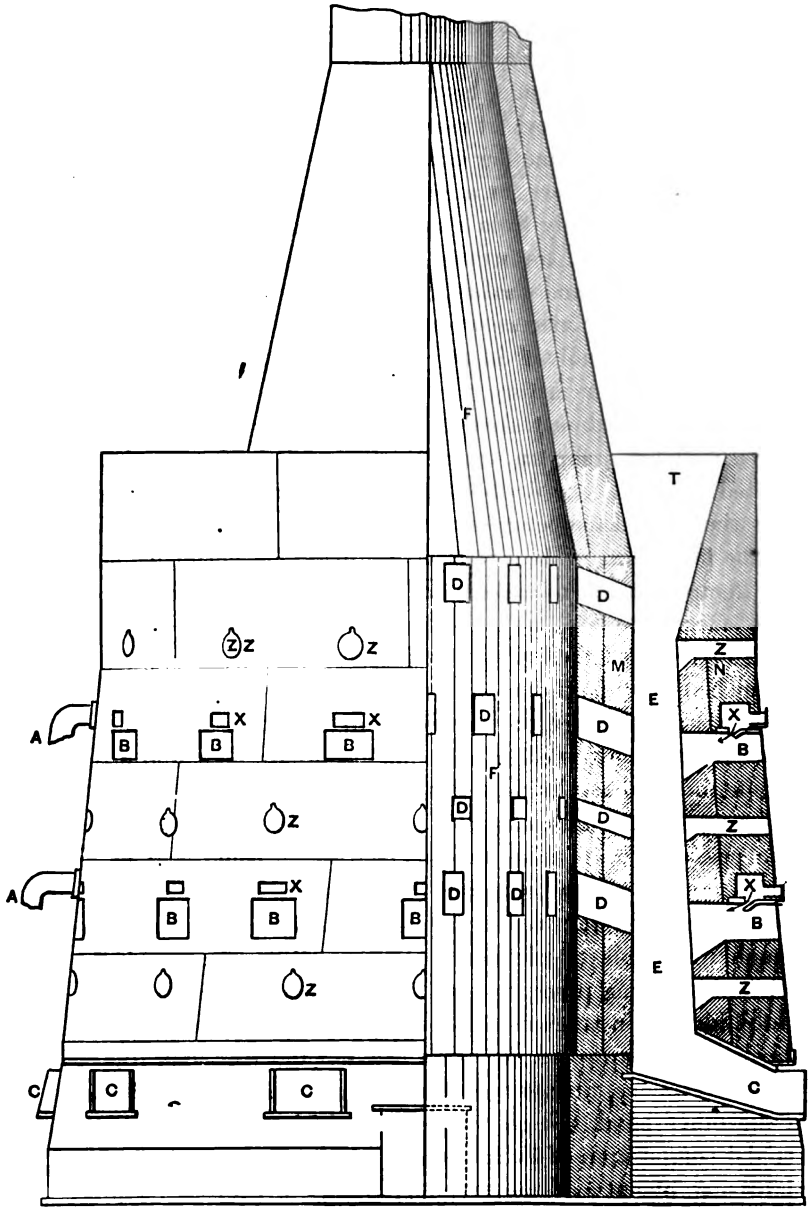

FIG. 2.—Davis-Colby Ore-Roaster, Cornwall Anthracite Furnaces, Cornwall, Pa. (Scale: $\frac{3}{16}$ inch to 1 foot.)

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sary for the removal of the second half of the sulphur. Heat is required in such a degree in each case as will most effectively enhance oxidation, *and as long as it is desirable to continue the oxidizing action, the heat should be kept at the most favorable point, with ample opportunity for admission, at the same time, of atmospheric oxygen.* It is this fact that brings out the peculiar effectiveness of the Davis-Colby roaster.

Some analyses are appended to show results obtained in actual practice with the Davis-Colby roaster.

Excellent results have been obtained in working on Katahdin, Me., ore, which is very open and porous, and well adapted to roasting. The following table shows its composition when raw:

Raw Katahdin Ore.

	I.	II.	III.	IV.	V.
Iron	47.75	57.34	44.80	39.50	53.50
Phosphorus	.017	.025	.04	.035	.034
Sulphur	3.90	1.29	.98	2.50	2.48

The roasted ore contains the following percentages of sulphur:

	1.	2.	3.	4.	5.	6.
Per cent. Sulphur, . .	0.07	0.15	0.10	0.18	0.12	0.10

The roasted ore runs, as a rule, from 0.05 to 0.20 per cent. of sulphur, averaging not over 0.10 to 0.12 per cent.

Cornwall, Pa., ore, containing on an average from 2½ to 3½ per cent. of sulphur, gives the following results when treated in Gjers kilns:

Cornwall Ore Roasted in Gjers Kilns.

Per cent. Total Sulphur.	Per cent. Sulphur as Sulphate.	Per cent. Sulphur as Sulphide.	Per cent. Total Sulphur as Sulphate.
1.133	0.283	0.850	24.977
1.380	0.135	1.245	9.782
1.873*	0.096	1.777	5.125
.800	.....	.....	.....
1.410	.....	.....	.....
1.050	.....	.....	.....
1.120	.....	.....	.....

* Samples of clinkered ore.

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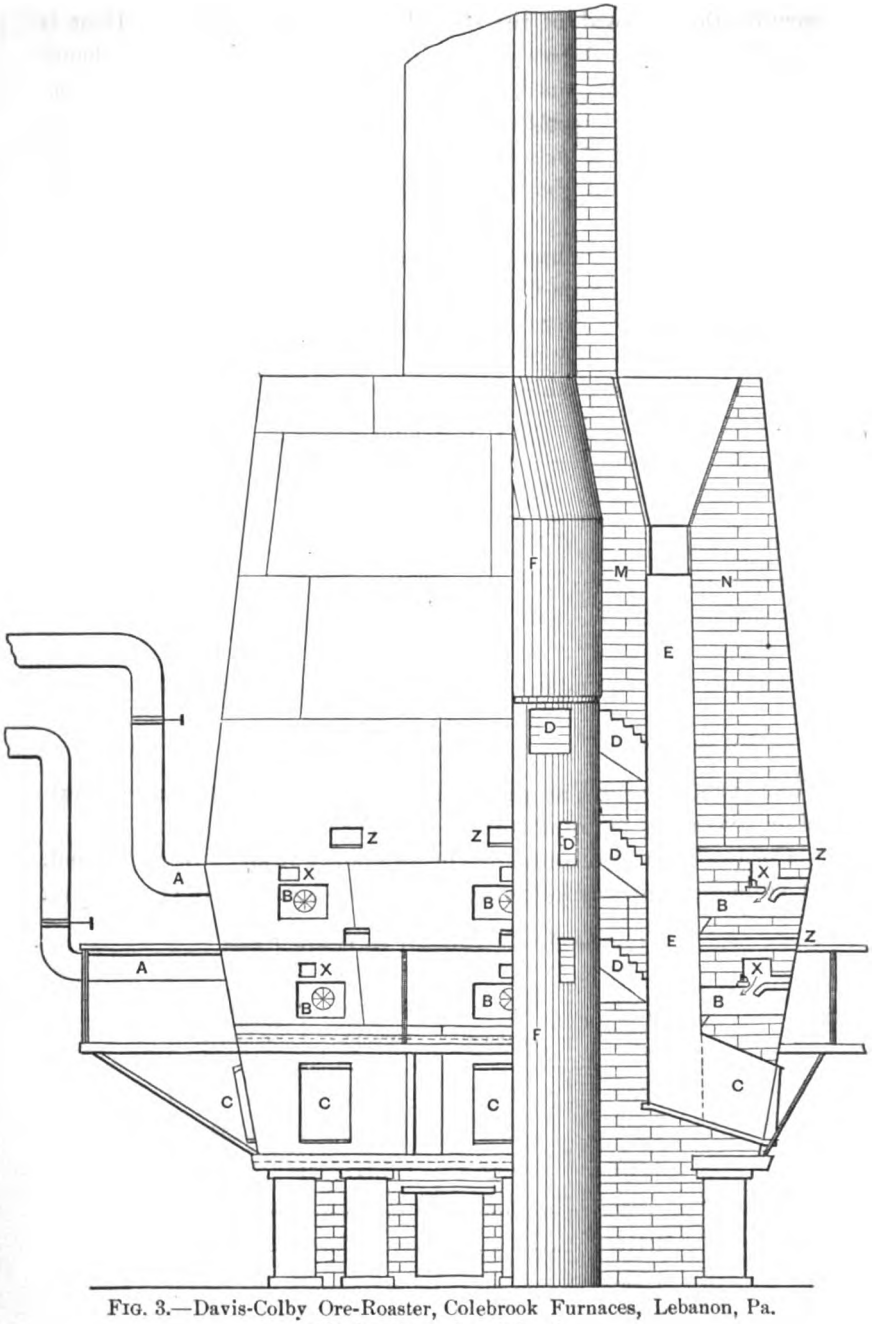

FIG. 3.—Davis-Colby Ore-Roaster, Colebrook Furnaces, Lebanon, Pa.
(Scale: $\frac{3}{16}$ inch to 1 foot.)

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The same ore roasted in the Davis-Colby kilns shows results as follows:

Cornwall Ore Roasted in Davis-Colby Kilns.

Per cent. Total Sulphur.	Per cent. Sulphur as Sulphide.	Per cent. Sulphur as Sulphate.	Per cent. Total Sulphur as Sulphate.
0.577	.....	.....	.....
0.779	.....	.....	.....
0.674	.....	.....	.....
0.666*	0.533	0.133	19.96
0.602	0.497	0.105	17.44
0.398	.....	.....	.....
0.697	.....	.....	.....
0.782	0.358	0.424	54.21
0.798	0.480	0.318	39.85
0.596	0 315	0.281	49.14
0.740	0.534	0.206	27.83
0.850	.....	.....	.....

As being of interest in this connection, the following results are given as obtained by another type of gas-fired roaster, the Taylor kiln, on magnetite ores from the Jones mine, Berks county, Pa. This ore is similar to Cornwall ore, and carries about the same amount of sulphur. The analyses are by Mr. R. H. Vondy, of the Phœnix Iron Company, and the roasting was done in their kiln:

"Jones" Ore Roasted in Taylor Kiln.

Per cent. Total Sulphur.	Per cent. Sulphur as Sulphide.	Per cent. Sulphur as Sulphate.	Per cent. Total Sulphur as Sulphate.
1.087	0.656	0.431	39.65
1.059	0.552	0.507	47.87
1.045	0.618	0.427	40.86
.912	0.548	0.364	39.91
.909	0.444	0.465	51.15

*Samples of clinkered ore.

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As to the ability of the Davis-Colby roaster to dispose of fine ore, it may be well to state that, when properly charged, as much as 30 to 40 per cent. of fine ore can be used. But fine and coarse should be thoroughly mixed in filling, as a solid mass of fine ore before a fire-arch chokes off the gas and prevents the heat and air from penetrating the ore. Kilns of the size shown in the illustrations, roast from 75 to 100 tons of a reasonably porous ore in twenty-four hours. Of course, a hard dense ore requires a longer exposure and the output of such ore is somewhat less. The roaster at the Katahdin furnace, 20 feet high and 15 feet in diameter, roasts about 40 tons per day, the entire supply for the furnace. The kilns at the Colebrook furnaces and Cornwall Anthracite furnaces, as shown in the illustrations, roast from 75 to 90 tons of Cornwall ore per day.

The occurrence of clinkers in this roaster cause little difficulty in working. They are local, seldom extending beyond the bounds of one or two fire-arches, and as the annular space is so narrow, and widens downward, they are readily reached and broken up while hot. Indeed, with proper attention, there is no reason why clinkers should be allowed to form at all, owing to the easy and full control of the heat. But carelessness of men in charge will sometimes bring about that result. The upper fire-arches should generally be kept at a somewhat lower temperature than the second set, gradually raising the ore to the desired heat as it descends, so that if clinkering does take place it is only after long heating at a lower temperature, and after atmospheric oxygen has had full play on the reasonably hot ore. It is seldom necessary to sinter an ore in roasting, but if it is done, it should be the final act of the process, as it renders further effective roasting impossible.

The cost per ton of roasting ore in this kiln varies with the circumstances, and it is scarcely possible to give figures. The cost is greatly affected by the method of breaking and filling the ore, and by the use of furnace-, or producer-gas.

New kilns with a capacity of 75 to 100 tons per day are built by contract for $3000, including royalty. But the system can be adapted to any ordinary Gjers kiln, the expense varying with the size and shape of the original roaster.

It is possible that this roaster may find a new field of work in the preparation of gold- and silver-bearing lead-ores, and other metallurgical processes of which roasting forms a part.

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NOTES ON SOME COALS IN WESTERN CANADA.

BY WM. HAMILTON MERRITT, F.G.S., ASSOC. R.S.M., TORONTO, CANADA.

(Ottawa Meeting, October, 1889.)

WITH the exception of the Vancouver Island coal, all the western coal-fields owe their present development to the completion, in the autumn of 1886, of the Canadian Pacific railroad. While it could not be expected that a very great deal could be accomplished in three years, enough has been done to pretty thoroughly establish the coal-bearing areas and their correspondence with those which have been developed to the south of the boundary along the lines of the transatlantic railroads in the United States.

This summer I visited some of the important developments in the coal areas of Washington Territory, largely with the object of being better able to appreciate the corresponding coal-bearing areas in British Columbia to the north.

In Western Canada, coal-bearing rocks have been found in three zones:

1. In the plains to the east of the Rocky Mountains and in the eastern flanking ranges, the coal occurs in the Cretaceous formation (including the Laramie).

2. In the interior plateau of British Columbia, the coal is found in the Tertiary formation.

3. On the coast of British Columbia, Cretaceous and Tertiary rocks are found carrying coal, and on the Island of Vancouver the well known Nanaimo coal has been worked for years in the first-named formation.

In all of these zones, the coals vary from lignites up to higher grades, the factor determining quality being the amount of pressure to which they have been subjected. The intensity of this pressure is generally shown by the disturbance which the coals exhibit, and, in many cases, is almost directly in proportion to the distance of the deposits from mountain ranges. This seems to be also the opinion expressed by Mr. Bailey Willis in connection with his Census Report on the coals of Washington. It has been elsewhere stated

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that super-imposed strata have been thought to have been an important factor in these changes; but my observations for several years in all these areas lead me to the conclusion that it is pressure alone from distortion and upheaval that has altered these western coals into the many varying grades in which they are found to exist.

In the first zone, an enormous amount of coal occurs in the territory between the western borders of Manitoba and the Rocky Mountains. I shall merely note some of the seams, which are reached by rail, as examples of the character of the coals in the area mentioned. In the plains they are all lignites, changing to a high-grade lignite at the Galt mines (which are reached from the Canadian Pacific railroad by a branch railroad 110 miles long), and to a bituminous coking coal at the Bow River mines (where a 7-foot seam cuts across the main line of the Canadian Pacific railroad). The maximum result of the metamorphic influence is reached in the Cascade Valley, where the pressure of the mountains, on both sides of the Cretaceous trough, has altered the coal which it contains into an anthracite.

The following analyses, passing from east to west, convey some idea of the types of these coals:

TABLE A.

Eastern Zone.

	a.	b.	c.	d.	e.
Water	20.54	10.35	6.50	4.41	0.71
Volatile combustible matter	33.26	34.40	38.04	40.32	10.79
Fixed carbon	41.15	39.61	47.91	48.27	80.93
Ash	5.05	15.64	7.55	7.00	7.57
Total	100.00	100.00	100.00	100.00	100.00
Coke	None.	None.	None.	Good.	None.
Approximate distance from Rocky mountains, miles	156	96	66	24	0

a.—Medicine Hat, lignite (Geological Survey, analysis by fast coking).
b.—Crowfoot, " " " " " "
c.—Galt, " " " " " "
d.—Bow River mines, bituminous, " " " "
e.—Cascade Valley, anthracite, " " " "

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In the interior plateau of British Columbia lignite and coal have as yet been found in only a few places. The following are the only occurrences yet discovered worthy of notice:

At Princeton or Allison's some 20 feet of alternating lignite and shale seams occur, lying at a gentle dip. The lignite can be obtained of a workable thickness, but the greater part of the bed is too much mixed with shale. The character of the lignite, as indicated by the analysis, is that of an inferior coal.

The lignite found at Marble Cañon, Hat Creek, is of a better description, as shown by the analysis. It is said to be of very considerable thickness. I did not think the quality sufficiently good to justify a visit to the place, which has been described in the Reports of the Geological Survey.

At Kamloops, close to the Canadian Pacific railroad, coal of a very fair bituminous character has been found; but as yet seams of only about a foot in thickness have been opened up. The vicinity is being tested by a shaft.

In the Nicola Valley, some 40 miles from the railroad, a seam of bituminous coal, about 5 feet in thickness, has been exposed. This coal has been subjected to a greater amount of metamorphic influence than any yet discovered in this zone. It lies adjacent to a mountain, which is probably a result of the disturbance that has altered it into a good coking bituminous coal.

TABLE B.

Interior Zone.

	a.	*b.*	*c.*	*d.*
Water	15.75	8.60	6.26	36.065
Volatile cumbustible matter	35.40	35.51	39.97	
Fixed carbon	41.45	46.84	48.22	61.290
Ash	7.40	9.05	5.55	2.645
Total	100.00	100.00	100.00	100.000
Coke	None.	None.	Fair.	Very good.

a.—Allison's, lignite.
b.—Hat Creek, lignite (Geological Survey).
c.—Kamloops, bituminous.
d.—Nicola, " (Geological Survey).

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On the Pacific Coast zone, on the main shore, there has as yet been located a very small amount of coal and lignite in the districts which correspond to the large areas developed along the Puget Sound, to the south of the international boundary. And, as has been ascertained to the south of the line, the coal which has been found near the coast is merely a lignite, but that which occurs inland, near the Cascade Range, has been altered into a bituminous coal. A sample of the latter type is found in a 2-foot, somewhat dirty, bed of coal, which has been opened on the slope of Sumas Mountain. Still further inland, the Cretaceous conglomerates occur near Chillawack, but all the coal which they have so far been found to contain consists of a few small masses forming part of the conglomerate, and some very thin strings of a coaly matter. The analysis from the above-mentioned Sumas Mountain seam is as follows:

Analysis of Sumas Mountain Coal.

	Per cent.
Water,	4.62
Volatile combustible matter,	35.68
Fixed carbon,	42.00
Ash,	17.70
Total,	100.00
Coke,	Fair.

THE MINING INDUSTRIES OF EASTERN QUEBEC.

BY R. W. ELLS, LL.D., GEOLOGICAL SURVEY, OTTAWA, CANADA.

(Ottawa Meeting, October, 1889.)

THAT portion of Quebec to which the few remarks I propose to make are more particularly intended to apply, viz., the Eastern Townships, has long been known for its mineral wealth, and has enjoyed a reputation for mining enterprises, second probably to no other part of Canada. Capital has been invested in large amounts at various points, some of which has yielded a handsome return to the investors, while in other cases the inevitable results of rash and foolish speculation have followed, much good money has been sent after bad, and the principal result has been a dearly bought expe-

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rience. This is a peculiarity which, I take it, is not entirely confined to Canada, and it can safely be said that this portion of Quebec does present to-day more advantageous openings for the investment of capital, wisely and skilfully applied, probably, than at any stage of its past history. It may be said, generally speaking, that these industries are at present confined to not more than half a dozen lines, among which may be enumerated, in the order of their present value, copper, asbestos, roofing and other slates, gold, the manufacture of lime. Silver, chromic-iron, antimony, nickel and the various ores of iron, have also been worked to some extent.

The relations between the geological structure of any country and its mineral wealth are very close and of the highest importance, but while, as officers of the Geological Survey, possibly the greater part of our energies have been devoted to the unravelling of the complicated problems of structure which have from time to time been presented in regard to the age of the several rock formations in Canada, a certain amount of attention has always been paid to the careful study of the economic aspect of the question. Some of the most intricate problems of geological structure have been encountered in that section of Quebec east of the St. Lawrence river, upon the elucidation of which much labor has been expended for more than forty years. It is no great cause of wonder, therefore, that changes of opinion have taken place concerning the age of certain portions, as new light has been presented by successive years of study.

To better understand the positions of the several mineral-bearing zones of eastern Quebec, it may briefly be stated that the rocks of that section are divisible into three grand classes, viz.: the crystalline schists, which occur in the form of extended anticlinal ridges, the slates and sandstones, often highly quartzose, which flank these ridges on either side, and the areas of volcanic rocks, diorites, granites, serpentines, etc., often of large extent, with which both the preceding divisions are intimately associated. As to their age, the views of those who have more recently studied these rocks differ widely from those expressed in the earlier publications of the Geological Survey. Instead of now regarding the great ridges of crystalline schists as altered Middle or Lower Silurian sediments, it has been very conclusively established, both from palæontological and stratigraphical evidence, that they are much older, being, in fact, for the most part, at least, pre-Cambrian, by which is meant that they underlie the lowest known fossiliferous Cambrian zone. In the same way much of what was forty years ago regarded as Upper Silurian,

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in which was included a very large area in the southeastern portion of the Province, has been found to really belong to the Cambrian and Cambro-Silurian systems. In this way a connection has been established with well-recognized mineral-bearing zones of other provinces and countries, in places where formerly a marked discrepancy in the relations of the rock formations was apparently presented.

As regards the geological position of the principal mineral-deposits of this section, we now find that the ores of copper, where found in workable quantity, are very largely confined to the areas of the crystalline or older schists. Deposits of considerable extent, it is true, are found at other horizons, more especially in connection with masses of dioritic rocks, but as a rule these have not, in so far as yet worked, proved very continuous or of marked value.

On the other hand, the asbestos, or, more properly speaking, chrysotile, occurs in serpentine rocks which are for the most part associated with the Cambrian slates and quartzose sandstones, although serpentinous rocks are found along with the schists at certain points. The roofing-slates, at one time regarded as of Upper Silurian age, are also now held to form part of the great Cambrian series, as is also the case, I believe, with the slates of the Welsh quarries, while the gold, the mining of which has, however, as yet been entirely confined to the alluvial deposits, or to the gravels of old river channels, is also without doubt in great part derived from quartz-veins in slates of the same Cambrian age. The statement as to the age of this last is important, from the fact that these slates were for many years regarded also as of Upper Silurian age, and have been so described in the *Geology of Canada*, 1863, in which the earliest views of the Survey as to the structure of this region, viz., those of 1847, are simply repeated. The resemblance of these gold-bearing slates, which are especially developed in the Chaudière and Ditton areas, to those of the Cambrian gold series of Nova Scotia was, however, pointed out many years ago, and referred to by Logan, Selwyn, Hunt and others.

The copper-deposits in eastern Quebec, operated for many years, and still worked, are those known as the Capelton mines. They are situated about seven miles south of Sherbrooke, the principal city of Quebec, east of the St. Lawrence, and are located on the eastern flank of a ridge of talcose, chloritic and micaceous schists, which in this locality, form a somewhat narrow belt, some miles in extent. It has, in the late reports, been described

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as the second or Sherbrooke anticlinal. Throughout this belt of schists, with which are associated considerable masses of greenstone or dioritic rock, deposits or veins of the sulphuret or copper pyrites are numerous; and a year ago, at the time of the great copper boom, a number of mines were opened in this section. While at many of these places the veins promised well at the time, the gradual decline in the price of copper, and the generally low grade of the ore, rarely yielding more than four to seven per cent., resulted in a gradual suspension of most of these enterprises. At the present time, while prospecting is again being resumed with some vigor, mining is confined almost entirely to two locations at Capelton, viz., the Nichols and the Eustis mines. The continued success of these workings has been due to several causes. One, probably, is the great size of the vein, which is stated to reach, in places, a thickness of 45 to 60 feet, and has been worked to a depth of over 1500 feet. A chief cause, however, is that these ores are rich in sulphur, the bulk of the raw ore being used, first of all, for the manufacture of sulphuric acid, the residue being subsequently utilized for the extraction of the copper. A certain percentage of silver in the ore is also, doubtless, a very important factor as regards their present successful working, the amount at times ranging as high as 10 to 12 ounces to the ton, and probably even higher, though probably 3 to 4 ounces represents the average.

The output of ore from these mines is large, ranging from 30,000 to 40,000 tons per year for each. While the greater part of this product is shipped in the raw state, smelting works, recently repaired by the Eustis Company, now dispose of over 1000 tons per month, which is reduced to matte on the spot. At the Nichols mine adjoining, a correspondingly large amount is used for the manufacture of acid. A portion of the latter, with the phosphate of the Ottawa Valley, is converted into superphosphate of lime. This industry, in the not distant future, bids fair to assume very large proportions. At the smelting works of the Eustis mine, the large percentage of sulphur is, at present, burned off and lost.

The larger lodes of copper-ore at these places appear to be mostly in the form of the yellow sulphuret or chalcopyrite, with which is found a very large percentage of iron pyrites. These have their largest development in the townships of Bolton, Hatley, and Ascot, though in some of the mines formerly worked, for instance, in the old Huntington mine, a large proportion of the pyrite is the magnetic variety or pyrrhotite. The rocks associated with these ores are,

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for the most part, schists of various kinds, with diorites and sometimes serpentine. Variegated and vitreous ores are also found, in considerable quantity, in the more westerly belt of Acton, Halifax and Leeds, where they have been worked to a very considerable extent. These are frequently associated with nacreous or talcose schists, as at Harvey Hill, or with limestones, slates and trappean rocks, as at Acton. At both of these localities very extensive workings have been carried on, and a large amount of very rich ore has been extracted. The mines of Harvey Hill are again being operated under a new management, the new company being styled the Excelsior Mining Company.

A very full description of the character and distribution of the various copper deposits of the eastern townships is given in the latter portion of the *Geology of Canada*, 1863, pp. 709–736, and supplemented in the general *Report*, 1866, pp. 34–44. From these it will be seen that many localities exist in this area other than those just mentioned, which are really deserving of careful attention at the present time as a source of supply for copper, both for smelting purposes and for the manufacture of sulphuric acid. The great and present value, however, of the copper-deposits above indicated is more particularly due to the fact that, in the more easterly or Ascot belt, the veins of pyrites which are, without doubt, in many places, of great extent, contain a very large percentage of sulphur, rivalling, in this respect, most of those yet known to occur in the copper-bearing belts of the Atlantic portion of the adjoining States. Their great importance as a source of supply for ores suitable for the manufacture of acid is just now beginning to be fully realized. It is, presumably not too much to say, that within the next five or ten years, great developments in this respect will take place throughout this section.

In the belt of crystalline schists, both to the north and south of Sherbrooke, recent developments indicate that possibly the heaviest and richest ore-veins have not yet been reached; and, in view of the new conditions which have arisen, much of this district would repay careful prospecting. Further to the north again, in Ascot and Garthby, new developments are at present being made that promise fine returns, and bid fair to equal, if indeed they do not surpass, the well-known mines of Capelton.

The asbestos industry, although of comparatively recent date, has, since its inception in 1878, rapidly increased in importance, and bids fair to speedily become the leading mining industry of the prov-

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ince. While the occurrence of fibrous serpentine or chrysotile in the Townships has been known for many years, and has been mentioned in the earlier reports of the Geological Survey, its true economic importance remained unrecognized, and it was not until the opening of the Quebec Central Railway across the serpentine belt of Thetford and Coleraine that any importance was attached to this peculiar mineral. Although occurring to some extent with the serpentines of the Laurentian district, north of the Ottawa River, its development in this direction has not as yet been sufficiently studied to warrant a clear expression of opinion as to the actual value of these areas, and the production of asbestos or chrysotile is as yet entirely confined in Canada to the Eastern Townships, and to the serpentine rocks which are at present regarded as forming a part of the volcanic belt of the Lower Cambrian system of Quebec.

This mineral differs entirely from the actinolite of central Ontario, and the asbestos of the Townships, also presents several points of difference when contrasted with that from the Laurentian district. This is seen in the greater softness, silkiness and tenacity of the fibre of much of the former, a property which gives it great value for spinning and for its working into the finer variety of textures to which it is now so extensively applied.

The stratified rocks associated with the serpentines of the Townships in which the workable asbestos occurs are generally slates of various colors, black, gray, green, and purple, with occasionally conglomerate and sandstones, which are often hard and highly quartzose. These are frequently associated with diorites, which often form large mountain masses, as at Orford, Ham, Broughton and other points, and which in texture are generally fine-grained, both massive and concretionary, and in color range through shades of green to brown.

In the province of Quebec the serpentine extend for many miles, in a series of interrupted outcrops from the boundary of Vermont, nearly to the extremity of the Gaspé peninsula. At several places, notably in Thetford and Coleraine, and in the Shickshock mountain-range of Gaspé it presents a large development. It is sometimes found in bands of a few yards in breadth, apparently interstratified with the slates and sandstones, and sometimes with diorites, in conjunction with which it forms knoll-like hills or elongated ridges of considerable extent. In many places a gradual transition from the diorites to the serpentines can be observed, and it seems very conclusively settled, both from observation in the field on a large

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scale and from analyses in the laboratory of the Survey, that much of the serpentine is an alteration-product from dioritic rocks, rich in olivine or some allied mineral. While generally massive, the rock has at times a slaty structure, and in many localities the mass is much shattered and broken. More especially is this latter feature noticeable in quarries or areas where dykes of diorite, but more generally of a whitish granite or granulite, cut the serpentine. This latter rock, which has generally a grayish or grayish-white color, consists for the most part of orthoclase feldspar and quartz, but at times contains an admixture of mica, classing with a true granite. Whatever may be the age of these whitish dykes, which frequently become masses of considerable extent, they certainly are newer than the serpentines which they traverse, as is evidenced clearly by the alteration and shattering of the mass traversed along the contact. The view is held by many of those engaged in mining asbestos, that the presence of these smaller dykes exercises a favorable influence on the production and character of the veins, in the same way as the action of diorite dykes upon copper or other metalliferous strata on the production of ore-deposits.

Among the principal areas of serpentine which are found at so many widely scattered points, the most easterly yet known is at a point called Mount Serpentine, about 10 miles up the Dartmouth river from its outlet in Gaspé basin. The serpentine is here associated with limestone and surrounded by strata of Devonian age. Small veins of asbestos are found in the rock, but not yet in quantity sufficient to be economically valuable. West of this the next observed is the great mass of Mount Albert, whence it extends west in a great ridge for some miles. This mass is known to contain veins of chromic-iron, and traces of asbestos have also been observed, but the area has never yet been carefully explored with a view to ascertain the presence of the mineral in quantity, owing largely to the present difficulty of access.

In Cranbourne and Ware, to the north of the Chaudière River and in the vicinity of that stream between the villages of St. Joseph and St. Francis, several small knolls are seen, in all of which small and irregular veins are visible, but apparently not in quantity sufficient to render them economically important, at least in so far as yet examined. Further to the southwest, in Broughton, Thetford, Coleraine, Wolfestow and Ham, a very great development of these rocks is observed, forming at times mountain-masses from 600 to 900 feet above the surrounding country level, and presenting very pecu-

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liar and boldly marked features in the landscape by their rugged outlines and curiously weathered surfaces. The large areas of this division terminate southward at a point termed Ham Mountain, a very prominent peak of diorite which marks the extremity of the ridge. In this great area, which we may style the central area, asbestos can be found at many points in small quantity, but at a comparatively few does it occur in quantity and quality sufficient to warrant the expenditure of much capital in its extraction.

The third area, regarding that of the Shickshocks as the first, begins near the village of Danville, and may be styled the south-western area. Thence it extends through Melbourne, Brompton, Orford, Bolton and Potton, in a series of disconnected hills, to the American boundary, beyond which the continuation of the serpentines can be traced into Vermont. In these areas, with the exception of the peculiar isolated knoll near Danville, the asbestos has, as yet, been observed in small quantity only, and generally of inferior quality. Large areas of soapstone are found at points throughout the area, and the associated diorites have a large development. It must, however, be said of this section, that considerable areas, whose outcrops can be seen along the roads which traverse the district, are concealed by a dense forest growth, and the true value of such portions must, for some considerable time, be largely conjectural. In fact, until the forest and soil are completely removed by the action of forest fires, as was the case at Black Lake and Thetford, the search for asbestos is likely to prove difficult and unsatisfactory. It is, however, very evident from the studies already made on this interesting group of rocks in Canada, that all serpentines are not equally productive—a fact very evident even in the heart of the great mining centers themselves, where large areas of the belt are made up of what is known as barren serpentine. As a general rule, however, the rock likely to prove asbestos-producing can be determined by certain peculiarities of texture, color or weathering.

At the Thetford mines, and in that portion of Coleraine lying to the northeast of Black Lake, certain conditions favorable to the production of asbestos appear to have prevailed, and have led to the formation of numerous veins, often of large size, which, in places, interlace the rock in all directions. These veins range in size from small threads to a width of 3 to 4 inches, and in rare cases even reach a thickness of over 6 inches. The quality of the fibre, however, varies even in these localities, and while much of it is soft,

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fine and silky, other portions are characterized by a harshness or stiffness which detracts greatly from its commercial value.

Several theories have been put forth as to the origin of the asbestos veins. From a careful study of many places throughout the belt, it would seem that the theory of segregation applies equally to these as to quartz veins or other mineral lodes. The fibre of the veins, unless distorted by pressure, is always found at right angles to the sides of the fissure, and frequently in veins of what are regarded as of large size, say from two and a half to four inches, the continuity of the fibre is broken by a dividing zone or comb, if one may use the expression, of grains of magnetic- or chromic-iron. In certain cases, also, the mass of the serpentine appears to lose color near the veins or in immediate contact with them, but this feature is not always observed.

The fissures or veins are not always regular or persistent for any very great extent. A small vein at the surface, of half an inch in thickness, may quickly enlarge to one of three inches or more, and, continuing, may die out entirely, while others come in on either side. They have much the aspect of the gash veins in slaty rocks, though there are many instances seen where the fibre maintains a tolerably uniform size for considerable distances.

The containing rocks show the presence of numerous faults, as in other mineral localities, but posssibly in the serpentine these are often more plainly marked. These faults throw the veins from side to side, and frequently are of sufficient extent to cut off entirely the working face of a highly productive area, the rock on the other side of the fissure being often entirely barren. The sides of the fault, in such cases, show extensive slickensides, and frequently have great sheets of coarse or woody-fibred, or imperfect asbestos, along the planes of fracture. Occasionally, pockets or small veins of chromic iron are found in close proximity to the asbestos.

Much of the asbestos found at the surface, or in the upper workings, is discolored. This is generally the case where the rock is shattered by the action of the intrusive dykes, or from some other cause, and the coloring is doubtless due to infiltration of water along the lines of jointing or fracture, becoming less as the rock grows more solid. The quality of the fibre in such cases improves as the mine becomes deeper. A somewhat similar remark may be made as to the quality of the fibre at the surface as compared with that from a lower level, but proceeding from a different cause. Owing to the destruction of the forest over the serpentine-knolls by extensive fires or by the

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action of the weather, the asbestos veins, which are exposed at the surface, have frequently a certain amount of harshness, which entirely disappears as the vein is opened. This peculiarity is probably due to the action of the fire, by which a certain proportion of the contained water of the mineral is dissipated; and in this connection, also, a very interesting study is afforded by the comparison of the output at the Thetford mines with that from certain areas in the Black Lake district. At the former, the fibre is remarkable for its soft, silky character, and the percentage of harsh fibre is comparatively small. The same is true of certain of the Black Lake mines, but at others, though many veins of good size are also found, a very much larger proportion of the fibre is harsh or brittle. The cause of this difference has long been a subject of speculation to those more immediately interested in the subject, some attributing it to a difference in level of the two localities—a theory, however, which can scarcely be maintained in view of the fact that much of the output from the Black Lake mines, from the higher levels, is of the best quality.

The presence, however, of large masses of the intrusive granulite near Black Lake, as contrasted with its almost entire absence at Thetford, where it is found generally in small dykes, leads to the inference that to this intrusion of heated matter may be largely due the lack of softness and silkiness in much of the Black Lake output. The presence of such great masses would have a tendency to dissipate a certain proportion of the contained water from the chrysotile, which is really a hydrous silicate of magnesia, in the same way as the action of the forest fires, though on a much more extended scale. The same effect can be produced by placing a piece of the mineral in the fire or on a sufficiently heated surface, when in a short time it will be found to have lost its softness to a large extent and become harsh and brittle.

The extraction of asbestos in the present stage of the industry may rather be termed quarrying than mining proper, the workings so far being all open cuts. The rock is blasted out, broken up, carried to the dumps and subsequently cobbed by hand, the latter being generally done by boys. The smaller veins, of a fourth to a half inch, are frequently left in the rock of the dump, as not in the present state of the industry repaying the cost of extraction. There is no doubt that with suitable machinery for crushing and cobbing, the greater part of most or all of the dumps, which have already reached a large size, would easily pay for working over, while the removal of the dumps

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themselves, from the fact of their covering much valuable ground, will very shortly become a necessity.

The workings at Thetford are at present confined principally to four companies, viz.: King Bros., The Bell Company, Irving, Johnston & Co. and Ross, Ward & Co. These are all situated on a small knoll of serpentine which rises to a height of 80 to 90 feet above the adjacent track of the Quebec Central railway which traverses the properties. The output from all these mines is practically the same in quality and length of fibre, though in the quarry of Irving, Johnston & Co., veins of extra width, nearly six and a half inches, have been found. This extra length of fibre does not, however, possess any special advantage, since it is generally broken by grains or partings of iron, and for practical purposes good clear fibre of two to three inches, or even less, is quite as valuable as any. In addition to the properties at this place above mentioned, new workings have lately been commenced by Mr. A. H. Murphy, now of Montreal, and by Luck & Mitchell, of Sherbrooke, both of which promise well. These are to the west of the railway, and at a lower level, and some extra work may be involved in the removal of a considerable covering of drift in places. The ground between the railway and the Thetford River, which here marks the western limit of the serpentine, has also been lately proved to some extent, and portions of it found to contain asbestos in satisfactory quantity, but the greater facilities for working the upper levels are such as to give them the preference for the present.

At Black Lake, four miles south of Thetford mines, four companies are also located. These are the Anglo-Canadian, the Frechette-Dauville, the Scottish-Canadian and the American Asbestos Company or Wertheim's, at all of which a large amount of work has been done. They are all at higher levels than the mines at Thetford, and the output, as already stated, is not, as a whole, equal in quality comparing the percentage of firsts produced. This will be seen from a statement of the relative percentages of the two districts, though this is again to a certain extent affected by the difference in grading practiced by the several companies. Thus from the Thetford area the percentage of firsts ranges from 50 to 70; seconds from 7 to 30; thirds, from 20 to 40. From the Black Lake district, the percentages are: firsts, 10 to 50; seconds, 25 to 30; thirds, 20 to 60.

The prices of asbestos are at present quoted at a considerable variation, due to the fact just mentioned that the output from the several mines is to a certain extent differently graded. Thus the seconds of

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one mine are quoted at $75 per ton, while from another the price of firsts is very little more. Extra quality of firsts, again, naturally commands a higher price than the ordinary run of the grade, but generally speaking it may be said that the scale of prices for the several grades of the output at the mines is as follows: Asbestos, No. 1, $80 to $110 per ton of 2000 pounds; No. 2, $50 to $75, and No. 3, $25 to $40. In addition to the above a certain amount of waste is obtained from all the mines and is sold at from $10 to $15 per ton.

Taking the scale of prices and the percentage of the higher grades, it is seen that at the present time, commercially speaking, the properties of Thétford are entitled to the first place. This is to some extent explained by the fact that a considerable proportion of the Black Lake output is off-color, the areas worked lying partly below the crest of the main serpentine-ridge and having a considerable covering of drift. At Thetford, owing to elevation of the knoll on which most of the mines are located, the loose or shattered rock, if it ever existed there, has been largely removed by glacial action, the surface of the rock being frequently polished and striated. It has always been expected that the Black Lake output would increase in value when the shattered surface rock was removed, and this expectation has, to a certain extent, been realized in so far as the workings have proceeded, the rock generally becoming more solid at greater depths, and the color and quality of the fibre improving. New workings along the face of the ridge at this place have also disclosed the presence of large veins of excellent material, and greatly superior to that from the lower works, yielding a large percentage of first quality. Here the action of the intrusive granulites has not been felt apparently to any considerable extent, and the conditions in this respect more closely resemble those prevailing at Thetford.

In estimating the values of asbestos properties, the question of the relative percentages of output should be carefully considered, since, while the number of tons produced by two areas might be approximately equal, and to the uninitiated the properties in consequence be nearly equal in value, the one from the large amount of firsts would be capable of paying very handsome dividends, while the other would require great economy of management and yield far less satisfactory returns.

In addition to the areas just described, several others are worthy of notice. The knoll of serpentine near Danville, of small extent, has been worked for some years by Mr. Jeffrey, of Richmond, with

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good results. The veins at this place are not large, seldom reaching two inches in width, but the quality of much of the fibre is excellent, and, though small, a considerable number of them produce a good amount of asbestos. The proportion of firsts from this mine is at present about 20 per cent., and of seconds 60 per cent., the remainder, of course, being thirds, thus approximating more closely the output of the mines at Black Lake.

A more curious deposit, however, is that known as the Broughton mine. The asbestos here is apparently confined to a single vein, having a hanging wall, in places, of soapstone. This vein was remarkable for its large size when first worked, being at times nearly or quite a foot thick, as well as for the great silkiness and softness of fibre. It had more the aspect of a well-defined vein, and has been worked for a distance of several hundreds of yards, and to a depth of about 70 feet. In the lower workings, however, it decreased rapidly in size, and at last split up into minute strings, rendering it worthless. Another peculiar feature observed at one of the mines near Coleraine station is the presence of mica, or a similar mineral, associated with the asbestos veins, or in close proximity to them, and occurring in irregular fissures in the rock, of several inches in width. The rock is much shattered, and the mica is not, in so far as yet observed, found in any of the other mines in this section of the country.

In many places the serpentine is intimately associated with masses of soapstone. In such cases, while occasional small and irregular veins of asbestos may be seen, in none of those examined has the quantity, with the exception of the vein at Broughton, just described, been found to be of much economic importance.

The slate quarries, of which a number have been worked to some extent, occur at various localities throughout the Eastern Townships, and are presumably all confined to the slaty portions of the Cambrian system. Several of these were started in the colored bands of what was formerly styled the Lauzon and Sillery divisions of the Quebec group, where red and green shades are common; others were located in what were regarded as slates of Upper Silurian age, the later investigations in the field having very conclusively shown however that their true position in the geological scale is much lower. At the present time, owing to various causes, the economic extraction of slate is confined to but two places, viz.: that of the New Rockland quarry, near Melbourne, and that near Danville. The output of the latter is almost entirely confined to school-slates, while that of the Rockland Company embraces roofing-slates, billiard-table tops,

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mantels, etc. The character of this slate is reported equal to that of the Welsh quarries.

An interesting feature at the former place, geologically speaking, is the contact of the slates with the serpentine, probably by a line of fault. A ready demand exists for the output from both these quarries, and a large force of men is employed. The New Rockland quarry especially has an excellent plant, with a magnificent water power, transmitted to the works by cable, and an outfit of cable or travelling derricks, which work to perfection. This quarry has at present a depth of 200 feet, and is connected with the line of the Grand Trunk railway, four miles distant, by a narrow-gauge road, by which a great saving in the cost of transportation is effected.

The lime works of the Dudswell Lime and Marble Company are well worthy of notice, both for their very considerable extent and for the great purity of the lime manufactured. They are located on a belt of limestone of Upper Silurian age, which is in places highly fossiliferous, and which rests on the one hand upon Cambrian and pre-Cambrian rocks, and on the other upon the Cambro-Silurian sediments of the great eastern area. In certain portions the limestone has been altered to a highly crystalline and compact marble, which, when polished, presents a very attractive appearance, the corals, of which much of the rock is composed, being well brought out. Other portions of the marble show an intricate banding of dark or blackish shades, alternating with yellowish tints, forming the variety known locally as "black and gold," and which has been very favorably compared with the celebrated Portor marble of Italy. Several thousands of dollars have already been spent in testing the value of this marble-deposit, and the rock, a short distance below the surface, appears quite solid, so that large and compact blocks can be readily obtained. The stone for burning is obtained from large ledges of a dark or grayish rock, in places also crystalline, which in the upper quarry presents a nearly vertical face of 80 to 90 feet. This is in close proximity to the kilns, of which there are 10 in all, 6 in the upper or older works and 4 in the lower and newer works, about a half mile distant. Each of these kilns is stated to have a daily capacity of 300 bushels of lime, so that the entire output is not far from 2500 to 3000 bushels daily. The lime manufactured is remarkable for its purity, the foreign matter in the rock averaging not more than 1 to 2 per cent., and has already obtained such a reputation for excellence that it controls the market throughout a large

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part of Quebec and Ontario, and through the adjoining States as well.

It would scarcely be doing justice to the mineral wealth of the Eastern Townships should we omit all mention of the gold-fields of this district. Discovered first in 1835 by a daughter of one of the *censitaires* of Mr. C. DeLery, in a small stream called the Touffe des Pins, a branch of the Chaudière river, that gentleman, on being made aware of the fact, easily confirmed it by examination of the locality, and obtained several pieces of considerable value. Subsequent explorations of this stream and its tributaries showed that the gold was scattered over a very considerable area, being, in fact, obtainable in greater or less quantity in nearly all the branches of the Chaudière, as well as in the bed of that stream where it was accessible, nearly from the boundary of Maine to that of the seigniory of St. Marie.

The limits of this gold-field were gradually extended, and it was found on examination, both on the part of the Geological Survey and by private individuals, that the metal could be obtained, by simple washing in the pan, from the sands and gravels of nearly every stream south from the Chaudière to New Hampshire and Vermont. The character of much of the gold over this area was so fine that the returns were not found to repay the labor of hand-washing, but in certain portions, notably in the upper Chaudière and its branches, and in the southeastern portion of the province, in the township of Ditton, much coarse gold existed, from which nuggets of large size were obtained, reaching a value in some cases of over $1000. The greater part of this coarse gold was water-worn, but occasional large pieces were found which were comparatively ragged and had quartz attached.

For some years the source from which this gold was derived was a question not clearly understood. It was first regarded as belonging to the quartz-veins which occurred in the crystalline schists of the Notre Dame range of hills, which traverse a considerable portion of the country in a northeasterly direction, the materials from their disintegration being spread over a considerable area to the south. The age of the rock underlying the area in which the gold was most commonly found was, at that time, regarded as Upper Silurian.

On the upper branches of the Chaudière, and on the Du Loup stream, heavy beds of gravel occur which were deemed worthy of a careful test as to their value as a source of gold; and a series of wash

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ings, made in 1851–52, under very disadvantageous circumstances, however, and with very ordinary appliances, showed a very fair margin of profit to operations conducted in the large way and with a proper expenditure of capital, many of the large areas of gravel being found sufficiently rich to warrant such an enterprise. Unfortunately, the only attempts of any considerable importance in this direction, either through lack of proper management or other causes, do not appear to have been so successful as anticipated; an experience, however, that should not be held as conclusively establishing the unprofitableness of the entire section.

The Ditton area, being entirely in private hands, has furnished no official returns from which the quantity of gold obtained from this district can be ascertained; though, from the coarse character of the gold there found, and the fact that certain persons have carried on mining continuously for many years, it is evident that much rich ground exists on the upper branches of the Salmon River. The appliances for mining and saving the gold appeared to be of the crudest kind, no provision existing for the extraction of the finer portion of the yield. Nuggets of $50 to $150 are reported from this place, and many quartz-veins traverse the black slates and sandstones which form the country rocks. Some of these are, without doubt, auriferous, from the fact that ragged gold in quartz is found in close proximity to them.

The largest operations in gold mining have been carried on in the Beauce section, and principally on a branch of the Chaudière River, known as the River Gilbert. Here, also, the mining was of a very rude kind; but though the returns for this area are largely incomplete, the official figures show that for 1877 to 1883 a very considerable quantity of gold was obtained, even under the very unfavorable conditions that prevailed. Much of the gold was very coarse, large nuggets being frequent. Owing to litigation and other causes, the mining on this stream has, to a large extent, been abandoned, though a large part of the old river channel, from which the greater part of the gold was obtained, yet remains untouched.

The presence of these old river channels in the tributaries of the Chaudière River is a very important feature in connection with the profitable extraction of gold in this locality. Indications of these are visible on the Famine River, the Pozer stream, the Gilbert, which has been worked to some extent and already referred to, the Des Plantes and the Millstream, where successful mining is being carried on at the present time. The bottoms of these channels,

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which are, in places, many feet below the present water-courses of the streams, contain several feet of well-defined river gravel cemented with sand and clay, in which the greater part of the coarse gold occurs, the upper portion being largely occupied with quicksands, gravel and clay. On the Millstream, also, there appears to be a great lack of appliances for saving the fine gold, the greater part of which is undoubtedly carried down the sluices and lost in the main stream.

A series of assays and examinations, conducted by Messrs. Hunt and Michel, and published in the *Geological Survey Report*, 1866, shows conclusively that many of the quartz-veins in the Chaudière district contain gold in paying quantity. The greater ease of working the gravels, however, and the present lack of capital, has, as yet, diverted attention from the working of these quartz-leads. This lack of capital is, to some extent, due doubtless to certain peculiar conditions of tenure of the lands which are not considered satisfactory to those contemplating investment in this direction. If this were put on a proper basis, and the country thrown open as a mining district, good results would certainly follow.

It will, however, probably be found that the most profitable fields of investment will be confined to the areas of Cambrian slates, and, judging from the fact that the largest nuggets have been obtained in close proximity to certain well-defined quartz-veins which occur in the vicinity of lines of anticlinal structure, the principle now generally admitted in the Nova Scotia gold-fields, that the paying leads are located near these anticlinal axes, will doubtless be found to apply to the gold-fields of eastern Quebec.

In this connection it may be mentioned that in their character the slates and hard sandstones of the Quebec gold-fields are very similar to those of Nova Scotia, and they in all probability belong to the same geological horizon. The southward extension of the Ditton gold-field should be looked for and found in that portion of New Hampshire adjoining and lying on the upper waters of the Connecticut River between the Hall and Indian streams. Since black slates and schistose sandstones have been reported by Prof. Hitchcock as there existing, precisely similar in lithological character to those of the Ditton area, and since the strike of the beds in both places is continuous, the valleys of the streams in that section should be also auriferous.

Of argentiferous galena, rich veins are known to exist, more especially in the Cambrian rocks of the Chaudière River, and upon the

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Du Loup, one of its principal branches from the east. Assays from some of these veins show a percentage of silver ranging from $30 to over $400 per ton. The veins are of good size, but the largest deposit, that of Risborough and Marlow, is at present accessible with difficulty. This will be shortly overcome by the proposed extension of the Quebec Central Railway, which is to cross this area. Then these properties should become very valuable.

Of the iron-ores it need only be said that deposits occur at various points, some of which are of great purity and excellence. The great drawback, at present, to their successful working is the lack of suitable fuel for smelting, wood for charcoal being generally too scarce in the immediate vicinity, while the geological formations in Quebec and Ontario are such as to render the chances of finding coal in either province an impossibility. Should the problem of cheap transportation of coal, either from the mines of Nova Scotia or from those of the United States, to some central point, such as Montreal or Sherbrooke, ever be solved, it is confidently anticipated that these ores will form no inconsiderable factor in the mineral wealth of eastern Quebec.

The deposit of antimony near Garthby has been known for many years, and beautiful specimens can be obtained there, but the lack of a market, in view of the occurrence of large veins of the mineral in Nova Scotia and New Brunswick, and the lack of facilities for extraction on the spot, have hitherto interfered with its successful development. This, also, is one of the problems that should be solved in the near future.

It will be seen from this sketch of some of the leading minerals in this portion of the Dominion, that the chances for the profitable investment of capital in several important lines are not few. Improved methods of mining, a more extended market, and, in the case of the gold, greater security of title to those desiring to invest with a view to scientific mining operations will do much to improve them. The great value of these several industries is only now beginning to be understood, and it can be safely predicted that before many years the mining industries of this section will take a much more prominent position than they have ever yet enjoyed, and will be regarded with increasing favor as a field for profitable investment by capitalists both at home and abroad.

Discussion.

R. G. Leckie, Sherbrooke, P. Q.: Dr. Ells has referred to the occurrence of gold in the Eastern Townships. Through the courtesy

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of Messrs. Ives and Pope I am in a position to submit to the Institute samples of the gold gathered during the last year at Ditton, when it was washed from the bed of a small stream, within the width of 100 feet. The rocks of that portion of the country have been described by Sir William Logan as Upper Silurian, but Dr. Ells's more recent observations class them as Cambrian, perhaps Lower Cambrian. They are intersected by veins of quartz, varying in width from 6 inches to 1 foot. At a distance of two miles from the locality where the gold exhibited has been collected a bunch of quartz was struck, from which about $600 worth of gold was extracted simply by hammering, even without a mortar. No further explorations have been made. What gold has been obtained has been collected by the crudest means, the farmers washing it out in the most primitive way, after their harvesting has been finished. Within a year it is probable that a systematic effort will be made to trace the source of the precious metal.

NOTES ON THE GEOLOGY OF THE DEKAAP GOLD-FIELDS IN THE TRANSVAAL.

BY W. H. FURLONGE, JOHANNESBURG, S. A. R.

(Washington Meeting, February, 1890.)

WHILE fulfilling professional engagements, my travels over this portion of the Transvaal have been quite extensive—always on horseback however, so that anything like a thorough investigation of the ground passed over has been necessarily impossible; still, the more minute examination of certain small areas, coupled with general observations, has enabled me to form some idea as to the prominent geological features of the country as a whole.

Anything like a complete or comprehensive description is indeed out of the question, as there are no proper maps of the country and the facilities for observation are limited. A large plateau, called the "High Veldt," stretches from east to west across the Transvaal. It is generally level, or gently rolling, and has an average elevation of 6000 feet above the sea; it is destitute of timber, and in fact greatly resembles the western prairies of North America. Rock-outcrops are not common in it. It terminates very abruptly to the east and northeast, the descent of 2000 or 3000 feet into the mountainous country that occupies its borders being made in a

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very short distance. These mountains extend in an easterly direction for a distance of 40 miles, when they again terminate, very abruptly, in an apparently flat region composed of marshes and sandy plains, sloping gently, but regularly, to the shores of the Indian Ocean, a distance of about 100 miles.

The gold-fields of this section of the country lie among these mountains, whose summits, however, rarely attain the elevation of the "High Veldt," while the valleys between them are from 2500 to 3000 feet above the sea.

The accompanying map is the best obtainable; and, though it is far from accurate, it will give an idea of the geology and topography of the district.

The gold-fields are provisionally sub-divided by the Government; but for the purposes of this paper I propose to ignore both political and geographical boundaries, and will therefore include in my description all the country from the eastern edge of the High Veldt to the coast plains, and, from north to south, the region from the northernmost edge of the DeKaap Valley, to nearly the center of Zwazieland.

In general, it may be said that these mountain ranges run in a northeast and southwest direction, though they vary in many instances considerably from that course. As a rule they form fairly continuous chains, though isolated peaks are not uncommon. They usually present a serrated appearance, and are essentially similar, though in miniature, to the Sierra Nevada mountains of California; differing from them in one particular, namely, that their slopes are steeper. This I account for by the absence of frost in this region. Rock-disintegration, and the consequent formation of a talus, is comparatively slow work; and what little rock has been loosened is washed away by the torrents of rain so prevalent during the wet season. Only an eye-witness can form an idea of the violence of these storms, or of the enormous amounts of material removed to lower levels during one of them. It is entirely owing to this cause that there is but little "surface" or "wash" throughout this part of the country. At such times the rivers, which have, perhaps, an average fall of 25 feet per mile, become raging torrents; and when the floods subside, it is found that the channels are pretty well cleaned out to bed-rock. As a consequence there are no deposits of gravel, as we find them in other parts of the world, and little or no chance for river, and absolutely none for hydraulic, alluvial mining. None of the rivers show the evidences of great an-

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tiquity; there are no cañons or gorges cut away, or, if they exist, they are insignificant; no terraces, no well-marked channels, and the approach to the river-beds is in almost all cases down a very easy gradient. Nor, so far as I can ascertain, have any ancient river-beds been discovered in the country.

I have diligently searched for, but failed to find, evidences of glacial action, phenomena with which I happen to be very familiar from my residence in the Lake Superior district and the country north of it.

Here, as in other gold-mining countries, are found large granitic areas and belts, flanked by schistose and shaly rocks, which contain the auriferous quartz-veins.

The granite, occupying as it does such an extensive area, may be described first. Its prevailing color is white, or rather light-red. The greater part of it is true granite, containing, however, but little mica, which, so far as I have observed, is invariably muscovite. The quartz seemingly always predominates, and the feldspar, though principally light-red (occasionally white) orthoclase, is, as a rule, accompanied by a large proportion of plagioclase (albite?). Very generally there occurs in this granite a little dark-green hornblende, and occasionally the rock passes gradually into, first, syenitic granite, and then true syenite.

These granitic rocks are often decomposed to a considerable depth over large areas. I wish particularly to draw attention to the following facts, which may have some bearing on the theory of rock-decay, as advanced by Professor T. Sterry Hunt. The granite here is decomposed only, either in the natural drainage-channels, *i.e.*, below "water-level," or in places where the surface- and rain-waters can at least remain in contact with it for a considerable time. Consequently it is only in flat places that the granite is found to be decomposed; while on the slopes of the hills bordering these areas, and to which this decayed granite can be traced in unbroken continuity, the rock has merely the ordinary weathered appearance. It is quite solid under the hammer; and, furthermore, instances are frequent where boulders, evidently *in situ*, microscopically undecomposed, rest on the same material, which, being constantly wet, is in a decayed state. In other words, though subaerial decay goes on here as elsewhere, but not so rapidly, the almost universal decomposition of these rocks must apparently be mainly, if not entirely, attributed to the agency of meteoric waters remaining for some length of time in contact with them.

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The orthoclase is usually transformed into an arenaceous clay, while the triclinic feldspars are more or less perfectly kaolinized, and sometimes crumble under the fingers. The quartz, of course, remains, though it generally soon disintegrates into a fine sand.

This decayed granite often covers immense tracts. The town of Barberton is built on the eastern edge of one of them, which is at least 16 by 18 miles in extent. A striking feature throughout the large areas, is what are locally known as "dongas." They are merely water-washes in the decomposed rock, are often of great extent and depth (the latter reaching 200 feet), and sometimes present the most fantastic and picturesque appearances. The accompanying photograph was taken of one of the more striking. I merely state, in explanation, that the rock seen is entirely decayed granite, which is on a "flat," and in a natural drainage-channel; while the hills in the background are composed of the same rock, but in a normal or solid condition. The fragments that have rolled down the hill on to the "flat," and which have been subjected only to the ordinary atmospheric action, are also undecayed.

This description of rock-decomposition not only applies to granite, but to all feldspathic and apparently eruptive rocks. With them, the decay is even more marked, as they contain no quartz, and, though perfectly solid as blocks on the surface, the underlying mass can generally be dug out with a spade to depths ranging from 50 to 100 feet, or perhaps more. The decayed granite, on the contrary, though it will sometimes crumble in the hand, yields only to the pick, as a rule. I have also observed one or two instances of decayed hornblendic schists, though, so far as I know, these are rare.

In describing the remaining rocks of the district, it is convenient to have a starting point; so I take the large basin on the south-eastern edge of which the town of Barberton is built.

This basin, which I have previously stated to be entirely floored by decayed granite, is surrounded on all sides by mountains 1500 to 3000 feet in height. This last-named elevation is only attained on the western side, where an abrupt ascent to the flat-lying rocks of the High Veldt, which rest immediately on the granite, takes us into an entirely distinct geological horizon. Surrounding this roughly circular basin on the other three sides, are the rocks amongst which the auriferous deposits of the DeKaap fields lie.

All these rocks are highly inclined. Their strike is in general tangential to the edge of the granite basin, and they dip, in all cases, away from it, at angles varying from 60° to 90°. It would appear

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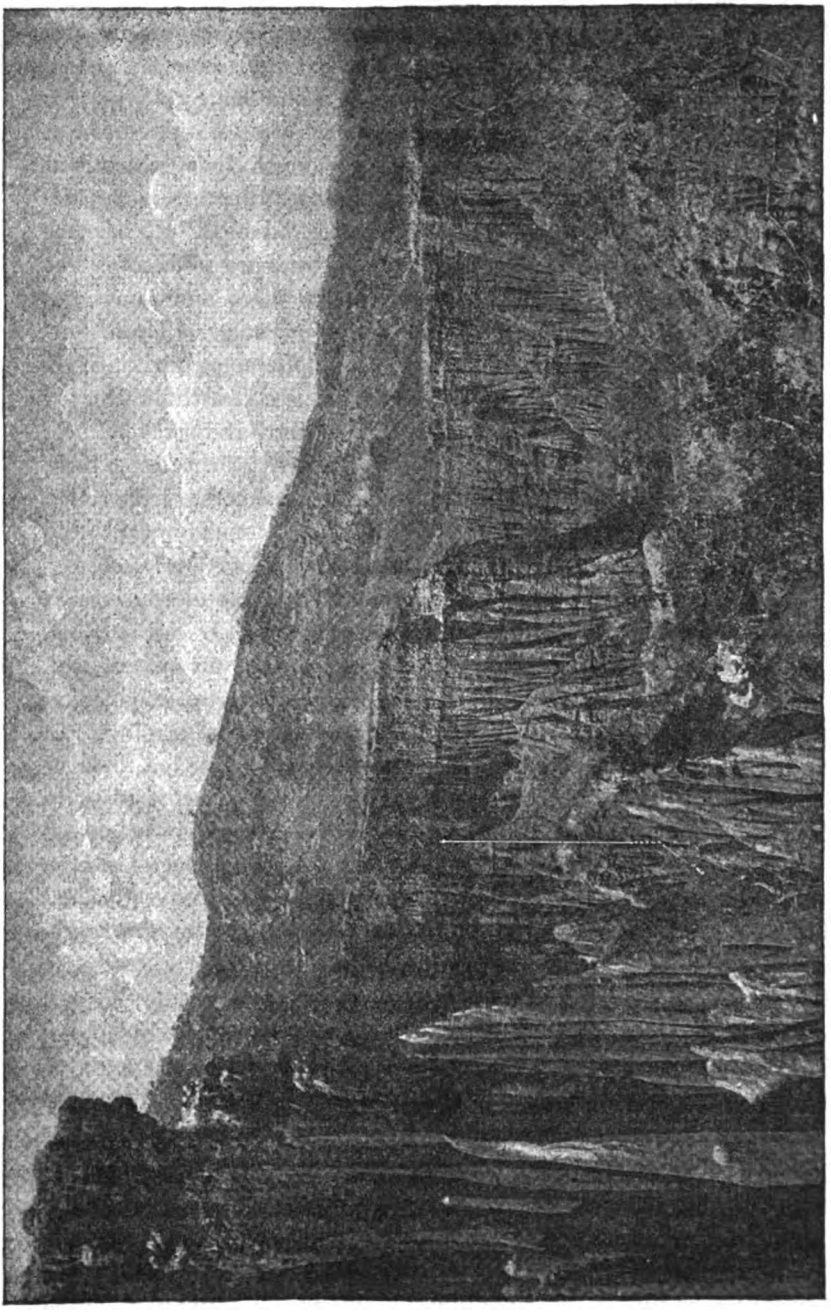

"Donga," produced by Erosion of Decayed Granite.

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that the rocks on the north and south sides finally come together to the east of the basin, while those which form the eastern border are apparently left as a kind of wedge, which rapidly thins out in that direction. How far east these rocks may go, I have, at present, no means of ascertaining.

The mountains forming the immediate northerly boundary of the DeKaap Valley are composed of schists dipping away from the decayed granite area, and they probably form a synclinal with the rocks, likewise schistose, flanking the still higher mountains further to the north, the mass of which consists of undecomposed granite. This belt of schistose rocks is, however, quite narrow.

Towards their western extremity, these inclined rocks are true schists, talcose, hornblendic, and chloritic; but to the east, they become more and more argillaceous, and lose their schistose structure, finally becoming true shales.

The rocks to the south of the basin, though in hand-specimens they might sometimes be termed schistose, are as a whole, shales; and they are not always argillaceous, but often distinctly hornblendic, and, I think, chloritic.

Interstratified with them, are beds of sandstone and conglomerate that are not found, so far as I know, on the north side. The sandstone consists of nearly pure quartz, and the conglomerate is made up exclusively of quartz pebbles, cemented together with a generally ferruginous sandstone, containing a small amount of talc.

To the south, these rocks extend well into Zwazieland, some 30 miles in a direct line, forming a succession of ridges, caused presumably by a series of foldings.

An important feature among these is, that there are several of these ridges which are entirely formed of silica. The mass of the rock composing them is apparently what may be termed a quartz-breccia, as angular fragments of white homogeneous quartz, dark amorphous quartz, and light colored pulverulent quartz are cemented together into a massive and compact rock, evidently by silica from solution; which, in the majority of cases, can now hardly be distinguished from the dark-blue fragments. It is deserving of notice, however, that the edges and angles of the enclosed fragments are quite sharp and well-defined; and that there is, as a rule, a noticeable quantity of the oxides of iron disseminated through the whole mass of the rock.

I have endeavored to ascertain roughly the thickness of this series,

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but want of time and of facilities for observation have prevented me. It must, however, be very great.

Though in general the strike of the tilted rocks is as I have described, there are many local variations which are very startling and which form special features well worth noting.

In other places, it is usual to find such rocks altering their strike gradually. Here they do so in most cases very abruptly. It is no uncommon occurrence to find, within a very few feet, these vertically-dipping rocks striking at right angles to their former direction, and, curious to relate, without any change in their angle of dip. In many places one can see these rocks which, from their composition, texture and color manifestly form part of the same deposit, abutting so squarely that one could almost believe that they had been cut as with a knife, and placed there; while at the contact they are perceptibly shattered.

I have never discovered true gneiss in this country, although I have occasionally observed the granite, especially in the vicinity of the flanking sedimentary strata, to become faintly foliated. But even then it could only be termed granitoid gneiss. Neither have I found, so far, any of the more uncommon minerals, such as schorl, topaz, apatite, etc.

The eruptive rock is evidently, geologically, the most recent. Its exact character has not yet been determined, though it is apparently a true diorite. Dykes of this rock are observed all over the country; breaking through, indiscriminately, the granite, the tilted schists, shales, and sandstones of the mountainous region described, as well as the flat-lying shales, sandstones and conglomerates of the High Veldt. The most remarkable thing about this rock is, that, though I have observed it over an area of more than 300 miles square, specimens from all parts are identical in character microscopically. The fact that it occurs as true dykes is most manifest in the granite; that rock having, by its decomposition, left the greenstone very prominently exposed. In the other rocks, however, the diorite (?) is generally seen on the surface as detached groups of rounded fragments, with no visible connection between them.

In most, if not all, cases, where the country is at all flat, this rock is, below the surface, decomposed into an unctuous red clay for a considerable depth; while the rounded surface-fragments, though perhaps decomposed more or less on their lower sides, where in contact with the moist decayed portions, are quite solid and sonorous

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under the hammer. In many cases (though I have never observed this in the granite) this rock has overflowed as a true lava, and in its downward course has involved fragments of quartz, sandstone or shale, which are perfectly angular.

In other places it is usual to find this rock amygdaloidal. The cavities are rarely empty, and, when filled, are always filled with secondary quartz, as are any cracks or crevices which originally existed in the rock. In these cases it is generally more or less schistose in character. With the exception of the amygdules, which are composed of quartz, this rock is perfectly free from that mineral, and the most patient investigation has failed to find any zeolites in the cavities. The hornblende, which usually occurs in circular crystals, largely preponderates over the feldspar, which is triclinic.

The wonderful similarity of this rock through the whole of this country, not only in its apparent composition, but in its mode of occurrence, its color, texture and the habit of its component crystals, makes it probable that the eruptions of these dykes were cotemporaneous. In any case, this rock has had a great influence, in some manner not understood, upon the auriferous nature of the neighboring veins and deposits.

There are few other plutonic or volcanic rocks to be met with. I have, however, on one occasion, seen a small area of porphyry, and, in two other instances, wide bands of foliated felsite, passing into eurite.

I have also observed belts, apparently interstratified with the formation, of amphibolite, containing small bunches of asbestos. It is, however, questionable whether this can be regarded as an eruptive rock.

A most remarkable feature is the almost entire absence of lime throughout the country. There are, I believe, deposits in the northern sections, but, so far as I can learn, they are very impure and consist of tuff. A small fragment from one of these, such as was being burnt for lime, did not effervesce with acid. A specimen of calcite was shown to me, from a long distance north, which was regarded as a curiosity. No dolomite, nor, in fact, any carbonates or sulphates seem to occur in this part of the country.

No less remarkable than the absence of lime is the large preponderance of silica, generally in the form of quartz, that is found not only in and amongst the shales and schists, but as secondary quartz, penetrating every crevice, crack and gas-pore in the eruptive or volcanic rocks.

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I have before mentioned how this mineral predominates as a constituent of the granite, but that is as nothing compared to the amount found amongst the slaty rocks. Here it occurs as segregated masses, interbedded with, and evidently in many instances replacing them to some extent. Indeed, these masses often assume such dimensions that they constitute the larger part of the whole country-rock. Their shape is generally irregular, sometimes angular, but in the majority of cases roughly lenticular. The lens-shaped bodies are generally connected—though this is not always apparent—by very thin seams or ribs of the same material of which they are composed. Or it may be that they are found to succeed one another at fairly regular intervals, either in a straight line or on both sides of, and at no great distance from, such a line, which can be termed the axis of their strike, and which coincides with that of the enclosing rocks. Many of these series of lenticular quartz-bodies are auriferous, and it is found that they are individually far more persistent in depth than in length. Still, the same peculiarities observed in following them along their line of strike are seen, perhaps not so clearly, on their dip.

Another class of interbedded quartz-deposits, which may be termed segregated quartz-veins, also occur in these rocks. These veins are found, as a rule, to extend for considerable distances along their line of strike. They are often very auriferous, and have, where worked upon, proved to be continuous to the limited depth attained up to the present time. Their width, however, is very variable. Since they are essentially similar to some of the California veins, I do not see why they should not be equally permanent.

These last-described veins, together with the large lenticular bodies of quartz, constitute one class of auriferous deposits of the DeKaap fields. They are generally composed of a dark-blue and homogeneous (never distinctly crystalline) quartz, which sometimes resembles quartzite, from its granulated appearance. There are, however, many instances where the quartz is milk-white, and it then shows, generally, some traces of crystallization. The associated minerals are all sulphides, and consist, usually, only of pyrite or pyrrhotite, more rarely of chalcopyrite and arsenopyrite. Galenite has been found in one or two places only, and sphalerite, so far as I am aware, has not been met with.

The gold, though more "free" near the surface than in depth, has nevertheless not all been liberated from the sulphides, for samples from some distance below "water-level," which are entirely

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undecomposed, often show large quantities of gold on being washed. The sulphides are, as a rule, very rich, and never, so far as I know, when free gold also occurs, contain less than 5 ounces of gold to the ton. Samples containing 30 to 50 ounces are not uncommon, while in one instance, I have found them to assay 760 ounces.

The shales or schists enclosing these veins are generally decomposed near the surface for some distance on either side, and are often found to be as rich, or even richer, in gold than the quartz itself. The decomposition of these rocks is evidently due to that of the pyrites, as below "water-level" auriferous pyrites are found to be disseminated through them for a limited distance on both sides of the veins.

Throughout these tilted rocks are found numerous interstratified beds of quartz-rock or quartzite. These are generally of large size, and, as a rule, are continuous for very long distances. Being harder than the enclosing rocks, and therefore offering better resistance to weathering influences, they stand out quite prominently, and are very easily followed. Locally, they are termed "bars," a name given to any persistent or conspicuous bed, a term which I shall adopt.

In spite of the fact that these "bars" are apparently interstratified beds, they usually split into a number of branches, which, though, perhaps, for a short distance diverging in strike from the main body, soon become approximately parallel to it, and without necessarily detracting from its width. Indeed, the branch may be, and often is, larger than the bar from which it originated.

The quartz composing these "bars" is never crystalline, but is usually compact and amorphous, from which state it may gradually pass into a roughly granular variety. In color, it varies from black to pure white. Very often the whole bar has a banded structure, ribs of dark and light quartz alternating, the dividing line corresponding to the strike of the enclosing rocks.

In my opinion, most of these "bars" are due to the replacing of the shales or schists by silica. Their structure, where banded, seemingly precludes any other hypothesis. In other cases, however, it is probable that they were originally strata of sandstone, which have been subsequently rendered homogeneous by solutions which permeated them, and which seemingly dissolved, to some extent, their component parts, and redeposited the silica in a gelatinous state.

Rain-water, in this country, seems to have more than usual solvent powers; as, in many places, I have observed the outcrop of sand-

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stone beds to consist externally of compact and amorphous silica, their real nature only being discovered by breaking up the rock. In these cases it is found that the ¼- or ⅛-inch of the crust, that has been exposed to the weather only, is homogeneous, the granules of the sandstone being easily distinguishable underneath.

The "bars" of quartzite have a very important practical value, inasmuch as the principal gold-deposits are found in, or immediately adjoining, them; though never, so far as I know, without there being also some eruptive rock in close proximity. There are apparently three ways in which the gold may occur in them:

First. In the body of the bar itself, where it is found in "chutes," or evenly disseminated, for a considerable distance, through the whole of it; in the latter case, more sparingly. The famous Sheba mine is an instance of the former mode of occurrence; and I could name other mines where the gold has been found more generally and widely distributed. It so happens that I have not had opportunity to study this kind of deposit.

Second. Where the "bars" themselves have evidently been fissured or fractured, presumably by volcanic outbursts, another variety of quartz has been deposited in the cracks and crevices therein formed. It often differs but little from that composing the bar itself, but, as a rule, is easily distinguishable from it, inasmuch as it is of a different color and texture, and more pyritiferous. The "bars" usually contain some of the latter mineral. These more recent deposits of quartz occasionally occur, more or less regularly, along one side or the other of the "bar" itself; but wherever this has been fractured, they penetrate into the body of it. More than one of our richest properties are being worked on this class of deposit, which, at 440 feet below their outcrops, show no sign of "pinching;" though the gold is, to a very great extent, locked up in the pyritous minerals.

Third. Often, instead of these deposits of auriferous quartz, in and alongside the "bars" there are others, of great extent, of iron oxides, containing from 30 to 40 per cent. of iron. I may safely say that all this ironstone carries gold, though not necessarily in paying quantities, or in a state to be caught by mercury. How it really occurs in these thoroughly oxidized ores is not known at present. At the surface the iron-ores consist generally of limonite and hematite, with some magnetite. In depth, however, the limonite disappears, and the ore then consists of hematite and magnetite, the former always predominating, while pyrite also is found in quantity. The gold con-

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tained in these ferruginous deposits varies from "traces" to 3 or 4 ounces per ton, and in the more extensive of the paying ones may be averaged at 18 to 20 dwts. per ton. The amount which can be extracted by ordinary milling and plate-amalgamation usually does not exceed 7 dwts.

The ferruginous deposits are not confined to the "bars," but are also found amongst the slaty rocks, where they are usually small, however, and are known as "burnt leaders" (from their color). These often "pan" well, and invariably assay better; but, so far as I have seen, they are neither large nor continuous enough to be worked with profit.

Some of the previously-described segregated quartz-veins are so ferruginous that they might fairly be entitled to be classed with these deposits, since it is manifest that these iron-ores did not originate in pyritous ores, as, in depth, they give place to a far more quartzose vein matter.

There is no experience from other parts of the world to guide us as to the probable depth to which these "bar"-deposits may continue; but the work now being prosecuted has at least proved, in some instances, that they are permanent to depths of 400 to 500 feet, and there is, seemingly, no likelihood of their giving out.

The segregated veins and series of lenticular-shaped bodies are, of course, just as likely to continue down as those in California and in other parts of the world; and, so far as opened, show no signs of "pinching," or of becoming poorer, wherever I have had opportunity of examining them. In fact, I am of opinion that the future of these fields, so far as gold-mining is concerned, is certain to be prosperous when the questions of cheap fuel and cheap transportation have been solved.

I must further mention, that many beds of argillaceous material, which enclose small "stringers" of quartz, and which, presumably from the decay of the previously-contained pyrite, are now a ferruginous red clay, contain paying amounts of gold.

Nor must I forget the beds of conglomerate which are enclosed in the tilted strata, and which, in a number of places, have been found, to a large extent, impregnated with pyrite. In some instances, the whole bed, when oxidized, shows by washing a paying amount of "free" gold; and, when pyritous, often assays over 1 ounce of gold to the ton. The auriferous pyrite, and, consequently, the "free" gold, are evidently of later date than the beds themselves, and neither of them, so far as I know, is ever found in the quartz peb-

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bles. They are entirely confined to the cementing material, which is, to a large extent, very porous. Specimens of pyrite in the pebbles themselves, where observed, occur invariably in cracks or crevices, into which they could have been infiltrated. Before very long I look for a large yield of gold from these beds.

The so-called "alluvial" of these fields is really not alluvial gold at all, in the proper sense of the term, and presents so many striking, and, as far as my experience goes, unique peculiarities, that it is well worthy of description.

This "alluvial" is usually found on the tops of rounded hills, and in a position where it could hardly have been placed mechanically. An examination of the ground almost always shows that the soil is simply a decomposed felsitic rock, probably a diorite in situ, which contains the nuggets. There is generally a large amount of quartz-fragments also, and seemingly the largest quantity of gold is found where these are most plentiful; but though the nuggets are sometimes found quite smooth and apparently waterworn, the quartz fragments are always angular; and, indeed, I have never observed a pebble anywhere throughout the whole area of auriferous ground. Moreover, the gold only occurs at or near the surface, say not deeper than 2 or 3 feet; and it is within the first 4 inches that all the larger nuggets have been found. It must also be observed, that though the masses and fragments of loose quartz have been, of course, repeatedly examined for gold, none, I believe, has ever been discovered in them by washing or otherwise; while, on the other hand, most of the large gold nuggets contain small pieces of quartz, which in appearance are identical with those found scattered over the surface. There can be but little question that these quartz fragments constituted originally the secondary quartz which filled the cracks and crevices of the more decomposed volcanic dykes.

Of course, a certain amount of the gold has found its way into the creeks and river-beds, where it occurs as true "alluvial," but it is so insignificant in quantity that I only mention its existence.

The so-called "alluvial" gold occurs also in rather considerable quantities, and under nearly similar conditions, amongst the flat-lying rocks of the High Veldt; but there the nuggets are found to be distinctly crystalline, and are far more often associated with quartz, though the latter has never been observed in situ containing gold as nuggets. The main fact of importance is, that it is there,

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as elsewhere, found exclusively in the surface-portions of the decayed feldspathic dykes.

Gold which is probably true, though ancient, "alluvial," is also found amongst the sandstone beds of the High Veldt; and nuggets of considerable size are frequently liberated by its disintegration.

I have before mentioned the white quartz occurring as veins and stringers in the crevices of the granite. It is worthy of note that, in some places, these contain an extraordinarily large percentage of gold. But as they are at best what in miners' parlance are termed "gash"-veins, they are of no special interest, except that since it is probable that not only the tilted rocks of the DeKaap fields, but also the flat-lying ones of the High Veldt, were derived directly from this granite, the presence of gold in it gives a possible clue to the origin of that found in the sandstone and, perhaps, the conglomerate beds.

There are among the schists and shales of this district enormous amounts of iron-ore. Besides the deposits hitherto mentioned as occurring in "bars," there are stratified beds, hundreds of feet thick, of almost pure hematite, together with some magnetite. These beds generally merge gradually into ferruginous sandstone, and then into true sandstone strata. At any rate, the Transvaal will never be at loss for iron-ores, some of which at least are Bessemer ores. Copper is not plentiful. It occurs as chalcopyrite in the veins, and the products of its decomposition are chiefly dioptase and chrysocolla and some other rarer silicates.

Lead is seldom found as galenite in these rocks. Among the High Veldt beds this mineral forms large deposits in places.

Zinc has never, so far as I know, been found in this country.

Antimony or stibnite I have observed in one or two places in unimportant quantities.

During my experience in this part I have never seen any silver-ores, though on the High Veldt some of the galenite contains amounts of this metal (up to 600 ounces to the ton) that may soon create a large silver-mining industry. The gold of these fields contains about 16 per cent. of silver.

Mercury, as cinnabar, has recently been found, although up to the present only in small quantities amongst the tilted rocks. Upon the High Veldt it has also been obtained. I have seen some very handsome hand-specimens of that mineral, which apparently promise well for the future.

Considerable quantities of nickel, and especially of cobalt, occur

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in these rocks to the north of the High Veldt; but the mines have been closed, owing to the cost of mining in this remote locality, together with the comparatively low price and small demand for these metals.

No salt, gypsum, apatite, or any of the non-metallic minerals have been discovered in quantities sufficient to be of economic value, except asbestos. I have never, personally, seen any of the asbestos deposits, but from samples I should not be surprised that when railway communication is established with the seaboard a profitable industry in that article of commerce will spring up.

The great drawback to this district is the want of fuel and timber. Of the latter, there is but little, and that is generally of no value for mining or building purposes, although in portions of Zwazieland there are "bushes" of fairly good timber suitable for these purposes. Coal, in inexhaustible quantities, is to be had on the High Veldt, within about 70 miles of these fields; but the only method of transportation being by ox-wagons, the price precludes its use for motive power.

Finally, it may be interesting to note, that the only traces of volcanic action are a number of hot springs, mostly giving off sulphuretted hydrogen.

ELECTRICAL ACCUMULATORS OR STORAGE-BATTERIES.

BY PEDRO G. SALOM, PHILADELPHIA, PA.

(Washington Meeting, February, 1890.)

THE broad general distinction between primary and secondary batteries lies in the fact that in the former the current is produced by the consumption of one or both of the elements composing the cell, while in the latter the elements retain their form as such, and merely pass from one state of chemical combination to another.

Much has been said in condemnation and but little in praise of storage-batteries; yet, notwithstanding the severe criticisms passed upon them, the skepticism with which electricians have viewed them, and the ignorance with which they have been handled, they continue to push their way into use. Their manufacture has now become a well-defined and established industry.

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Speaking of this subject, Prof. W. E. Ayrton, in a recent lecture, says: "Secondary batteries have caused much heartburning, for their uses, from the apparent fickleness of their complex chemical action, are yet but imperfectly understood. But we have at length been taught what is good and what is bad treatment for them, and after years of brave, persevering application on the part of the Electrical Power Storage Company, that forlorn hope, the secondary battery, has become one of the most useful tools of electrical engineers."

History.

The evolution of the storage-battery to its present form (which we must admit is not by any means perfect) has been painfully slow and tedious. A history of the various attempts and failures, even up to the time of Planté, in 1859, would fill several volumes, and would be entirely beyond the scope of this paper. It will suffice, therefore, to say that the germ of the secondary battery dates from the discovery by Gautherot, in 1801, that the plates of a voltameter give a reverse current. It was thought at first that the plates had been charged in a manner somewhat similar to the charging of a Leyden jar; but this idea was, after a time, demolished by the chemists, who showed that a definite chemical reaction had taken place, and that the return current was due to the chemical action of the new substances produced by the original decomposition. From 1801 to 1859 the subject was studied by the most eminent chemists and physicists of their time, viz., Ritter, Grove, Oersted, Faraday, Becquerel, Schoenbein, Wheatstone, De la Rive, Joule and many others.

In 1859 Gaston Planté, who made the most exhaustive researches in secondary batteries, constructed a secondary pile which is really the parent of all modern accumulators. A number of attempts have been made since that time to store the electrical energy of a dynamo, the most notable of which are those of Messrs. Thomson & Houston, in 1879, who patented and afterwards exhibited at the Franklin Institute, in Philadelphia, a gravity form of the Daniell cell, in which the zinc was reduced from its solution and the copper re-dissolved by the action of charging the cell from a dynamo or other source of electricity. C. B. Brush, of electric light fame, M. Faure and M. Julien are the last in the field with their respective inventions.

Theory.

The standard Julien accumulator is that known as type "B." It consists of nineteen plates, nine positive and ten negative, weighing

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25 pounds net, *i.e.*, without the box containing them or the acid solution. These add to the weight from 5 to 7 pounds. The following are the principal details:

Floor space, $6\frac{1}{2}$ x $5\frac{3}{4}$ inches.
Height, 8 inches.
Net weight (of plates), 25 pounds.
Total weight, 32 pounds.
Quantity of acid solution, about $1\frac{3}{4}$ quarts.
Strength of acid solution, 23° Beaumé.
Specific gravity, 1.185.
Electro-motive force, 2 volts.
Internal resistance, .005 ohm.
Charging currents, up to 20 amperes.
Discharging currents, up to 30 amperes.
Current capacity of each accumulator 150 ampere-hours, at a 10-ampere rate.
Energy capacity, 300-volt ampere-hours or watt-hours.

Each plate consists of a grid about 6 inches square and $\frac{1}{8}$ inch thick, cast in iron moulds. It is composed of an alloy of lead (95 per cent.), antimony and mercury. The square holes or interstices are filled with litharge and red lead for the negative and positive plates respectively. These materials, mechanically applied to the grid or support-plate, which is in reality nothing but a conductor, constitute what is known as the active matter. They become more or less sulphated when placed in the electrolyte, and the active matter is changed by the action of the charging current, as explained below, into reduced or spongy lead and peroxide.

Of course, it is hardly necessary to say that there is no actual storage of electricity in an accumulator; it is simply a convenient reservoir for storing the active ingredients necessary to generate a current of electricity. In fact, a charged battery becomes a primary battery, and the current derived from it is based on identical principles.

The reaction which takes place when a battery is discharged can best be examined by referring first to a primary battery with a simple zinc-copper couple. Mr. John T. Sprague has pointed out that the old explanation of the working of a battery by the decomposition of water is not the correct one.

When a positive element like zinc is opposed to a negative element in dilute sulphuric acid, the following reaction occurs: On uniting

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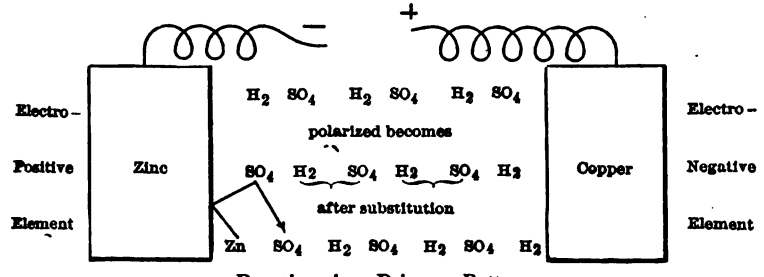
the poles the molecules of sulphuric acid become polarized, then the acid radical is turned towards the electro-positive element, zinc, and the basic ion towards the electro-negative element. Substitution then takes place, and the sulphuric radical unites with the zinc to form a molecule of zinc sulphate, and an interchange of molecules takes place along the line until the negative pole is reached, where hydrogen (two atoms) is liberated.

This can be expressed graphically as follows:

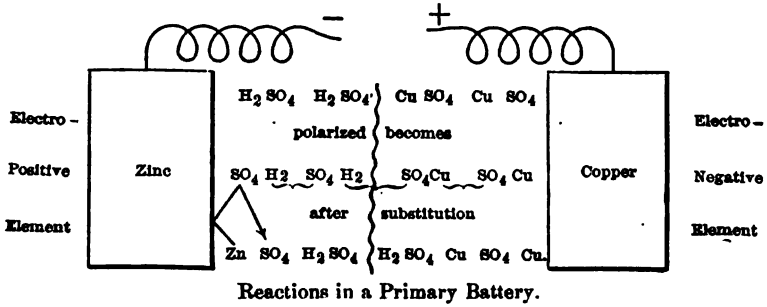
FIG. 1.



Reactions in a Primary Battery.
(Electrolyte, dilute sulphuric acid).

Let it be supposed that at the electro-negative element, where the hydrogen is liberated, there is some substance that it can combine with or displace, then the following reaction takes place, as in the ordinary Daniell cell:

FIG. 2.



Reactions in a Primary Battery.
(Electrolyte, dilute sulphuric acid, and copper sulphate solution).

The first action is the polarization of the molecules; the second a general substitution all along the line, forming zinc sulphate at one end and depositing metallic copper at the other, one atom of copper being equivalent to two atoms of hydrogen displaced in the former reaction.

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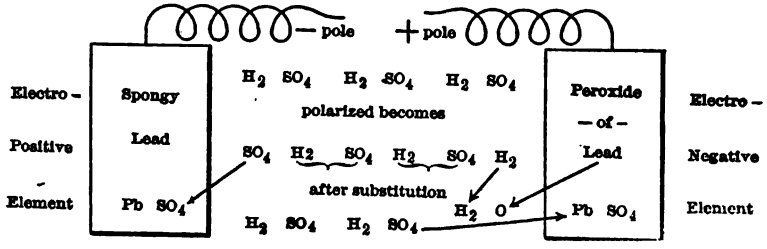
With the knowledge thus gained, we can now study the discharge of a secondary battery:

FIG. 3.



The spongy lead is gradually converted into lead sulphate.

The peroxide, giving up one atom of oxygen, is reduced to protoxide, and, in presence of sulphuric acid, converted into lead sulphate.

Reactions in the Discharge of a Secondary Battery.

In this case the polarization occurs as above, and the substitution then takes place, forming a sulphate of lead on one pole and liberated hydrogen at the other. But here there is a substance, peroxide of lead, which yields up its oxygen to the liberated hydrogen and forms a molecule of water. The oxide of lead formed by the reduction of the peroxide is, of course, instantly changed into sulphate of lead by the sulphuric acid. The action of the discharge is therefore to gradually sulphate both poles of the battery. Sulphate of lead, being insoluble in sulphuric acid and water, remains in the grids or support-plates in place of lead and peroxide.

The reaction in charging is more complicated and is not so well understood; but the final result is to leave the plates in the same condition as mentioned above before discharging. The conditions in charging, moreover, are not identical with those of discharging. It has been found that when a current from an external source is passed through two platinum-plates in acid, polarization results, and oxygen is liberated at one pole and hydrogen at the other.

When a secondary battery is completely discharged the plates are practically similar, that is, they are both lead-plates, containing sulphate of lead. Hence, there is no difference of potential, and no current results from connecting their terminals. It is therefore immaterial (so far as the chemical action is concerned) to which pole is connected the positive pole of the external source, which may be either a battery or a dynamo. In either case the plates now act in a manner similar to the platinum-plates mentioned above, and oxygen is liber-

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ated at the pole through which the current enters and hydrogen at the opposite pole. The reaction at the positive pole is:

$$PbSO_4 + O + H_2O = PbO_2 + H_2SO_4.$$

At the negative pole it is:

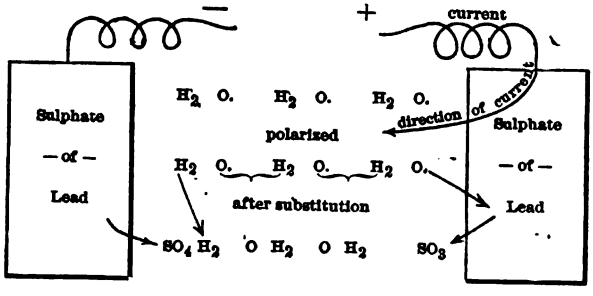
$$PbSO_4 + H_2 = Pb + H_2SO_4.$$

In other words, peroxide of lead is re-formed at one pole, spongy lead at the other, and sulphuric acid at both poles. The battery is then once more in condition to perform work.

The reaction can be represented graphically as follows (see Fig. 4), the current flowing in the opposite direction from that shown above.

That sulphuric acid is formed at both poles is proved by the specific gravity of the solution after charging. What has been done, then,

FIG. 4.



Becoming spongy lead and forming sulphuric acid.

Becoming peroxide and forming sulphuric acid.

Reactions in Recharging a Discharged Secondary Battery.

when a secondary battery is discharged, is to burn lead to sulphate. The amount of energy produced thereby can be determined quantitatively.

It will be necessary, before proceeding with the calculations, to go a little into molecular physics.

The mechanical equivalent of heat was the direct result of the true theory of combustion as established by the experiments of Lavoisier, Priestley, Scheele and others.

The mechanical equivalent of electricity has been determined only within the last few years, and its determination is founded solely on a chemical basis.

Unit E. M. F., or one volt, acting through unit resistance, one

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ohm, gives a current of one ampere, which, continued for one second, yields the unit of energy, one coulomb, equal to $\frac{1}{746}$ horse-power; and since one horse-power equals 550 foot-pounds of work per second, one coulomb equals $\frac{550}{746} = 0.7373$ foot-pounds in a second. Electric current is measured by the amount of chemical work performed in a given time.

Taking hydrogen as the unit, the amounts of the other elements set free by the same electric current are proportional to the atomic weights of the various elements compared with hydrogen as one, due allowance being made for the valency of the element under consideration.

Mr. John T. Sprague, in his great work on electricity, has proposed the name "Chemic" as the unit of electric quantity which will set free from combination one-grain weight of hydrogen in ten hours. It is possible from the above to calculate the amount of electric energy to be derived from a given weight of zinc or lead in a battery in the same manner as the amount of energy, and consequently the amount of work, to be derived from the combustion of a pound of coal under a boiler is determined. The "Chemic" is equal to 4673 foot-pounds.

The amount of current which will release .00001022 gramme of hydrogen (equals 0.000158 grain) per second is called the ampere-second or coulomb, which is the purely arbitrary unit of current. It is a great misfortune for scientific workers that it is so, and a matter of such importance that it may yet be changed. It is for this reason that Sprague adopted the unit which he calls the Chemic, mentioned above.

If the ampere had been based on hydrogen as the unit (one gramme, one grain, one ton, or one anything), then the electric equivalents of all the other elements would have followed in the order of their combining weights.

Sprague says: "Let us consider the weight as taken in grains; then taking hydrogen, one grain and one valency, as our base, we have the unit of quantity as that necessary to set free from combination one-grain weight of hydrogen or one atom of any monad, or to do equivalent work in any given atom; thus, iron 56 grains or zinc 65.2 (both dyads), would require two units of quantity to set them free, or would furnish two units themselves while dissolving. To convert this into current, time has to be taken into account, and for convenience of calculation I take ten hours."

The energy derived from the substitution of lead for hydrogen in

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sulphuric acid in the above reactions is the source of the E. M. F., and the amount or actual weight of lead consumed represents the current which will be produced, measured in coulombs.

Now, the atomic weight of lead is 207. Lead being a dyad, its electrical equivalent would therefore be 103.5; and $103.5 \times .00001022 = .00105800$ gramme. In other words, the consumption of .00105800 gramme of lead in a battery furnishes a current of electricity of one ampere for one second, or one coulomb.

One gramme of lead, therefore, equals 944 coulombs ($\frac{1}{.001058} = 944$).

In an accumulator consisting of nineteen plates, ten negatives and nine positives, we have in the ten negative plates about 5 pounds of spongy lead. Since 5 pounds = 2273 grammes of lead, 2273×944 (the number of coulombs per gramme) equals 2,145,712 coulombs. Dividing by 3600 (number of seconds in an hour) we have 596, say 600, ampere-hours at 1 volt pressure as the theoretical capacity of the accumulator, or about $\frac{8}{10}$ of a horse-power hour. This corresponds fairly well with actual practice in the total discharge of an accumulator, as given below.

This is a matter of the first importance, inasmuch as it sets at rest forever the extravagant claims and pretensions of those who, having only a partial knowledge of the subject, are very apt to claim more for their ideas than can be realized in practice. For every pound of active matter in the form of spongy lead, 120-volt ampere-hours is the maximum output that could conceivably be obtained. If 60-volt ampere-hours is reached, the results are equal to the best practice.

I am not able to calculate the theoretical E. M. F., as I have not the data at hand for the purpose; but, given the calories or foot-pounds per grain equivalent for the energy generated by the combination of lead and oxygen and oxide of lead and sulphuric acid, and dividing this number by 4673, the energy in volts represented by this reaction is obtained.

4673 (the equivolt of Sprague) is the mechanical equivalent of electricity in foot-pounds, *i.e.*, 4673 foot-pounds per volt equivalent, and he gives, therefore, the following definition:

"The E. M. F. corresponding to any chemical action is equal to the product of the equivalent weight of the basic ion, and of the heat of combination of the two ions expressed as C. G. S. units of work."

The above theory at least offers a simple and apparently intelligible explanation of the phenomena involved, and one also appa-

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rently in harmony with the facts. It will be of great aid in guiding our steps to further discoveries in the future.

Referring to the chemical equations once more, it will be noted that the function of the electro-negative element, or so-called positive plate, is to take up or give out, as the case may be, the atoms of oxygen liberated during the charge and discharge, and on its ability to do this depends the proper working of the battery. Lead itself, under the ordinary conditions of temperature, etc., having a small affinity for the sulphuric acid radical, the action would cease altogether if the basic ion of the acid or hydrogen did not have a constant source of oxygen to combine with. It is, therefore, a prime essential, in order to keep a battery in good working order, that the positive plates should not be allowed to sulphate, since in such a case the basic sulphate of lead formed is so difficult to convert into peroxide that it frequently happens that the grid itself is oxidized, in preference to the sulphate, by the action of the charging-current. When this action once begins, it is only a question of a few weeks as to the life of the battery, because the cell then becomes simply a Planté cell, in which the current oxidizes the support-plate. As this oxidation increases with each successive charge and discharge, it is only a question of time when the whole grid or support-plate will be oxidized, and the internal resistance of the battery so increased by the absence of any metallic conductor for the current that little or no current flows.

It is its freedom from liability to this defect which constitutes the excellence of the Julien battery. The resistance to oxidation of the triple alloy of lead, antimony and mercury is such, under the ordinary working conditions of the battery, that with proper care the active matter actually wears away by attrition of the liquid, and leaves the grid a solid metallic conductor. I have seen such grids after two years of active service in car-lighting, and they appear to be just as good as when first cast, and I am engaged now in experiments as to refilling them for active service again. The importance of this freedom from oxidation has not been fully appreciated; otherwise inventors and manufacturers of new storage-batteries would avoid a serious source of error.

Of course, if a battery is shamefully abused and neglected, it is bound to disintegrate, like any other piece of machinery or apparatus, no matter whose make or design it may be. The positive elemen or negative plate generally takes care of itself, and keeps in fairly good condition long after the positive plates have been ruined.

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The actual capacity of a cell depends on a variety of conditions of size, shape, thickness, rate of discharge, etc. Various designs of plates have been made to secure maximum capacity. Other things being equal, the thinner the plate and the greater the surface the greater will be the capacity in ampere-hours per given weight. The strength of materials and the space occupied are the limitations in the design.

An ideal plate would be one of great surface and exceeding thinness; but if an attempt were made to produce such a plate it would buckle by its own weight and cause trouble. On the other hand, a thick plate implies an immense amount of dead weight in the conductor; and since the active matter is so thick, and consequently the inner portions are more or less protected from the outside, the chemical reactions cannot take place readily. As a consequence, much of the so-called active matter is really inactive.

We found, after making a great many types, that what we call our 19 "B" is about the best adapted for general work either in lighting or in traction. Of course, there are conditions where it is desirable to use heavier currents than can be obtained with safety from so small a cell, but in such cases I prefer to use a greater number of plates in a cell, or to discharge them in parallel.

The commercial capacity of our standard 19 "B" cell is 150 ampere-hours at 2 volts or 300-volt ampere-hours or watt-hours. This is just about 50 per cent. of its *theoretical* capacity. Under favorable conditions we have had a commercial yield of 200 ampere-hours, or 66 per cent. of the *theoretical* capacity. This must not be understood as referring to a given charge and discharge. Of course, in a cell yielding only 150 ampere-hours we do not charge more than that amount. This capacity is from $\frac{1}{3}$ to $\frac{1}{2}$ greater than that of the same weight of any other cell manufactured, unless it is of a very recent date, the official tests of which have not been published in the electrical journals.

Prof. Gerard, of the University of Liége, made the following test with 24 elements or cells of the Julien type:

Charge.

Duration, 7 hours 33 minutes.
E. M. F. per cell, 2.35 volts.
Average current per kilo., 1.86 amperes.
Energy absorbed per kilo., 10,700 kilogrammeters.
Ampere-hours per kilo., 14.

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Discharge.

Duration, 6 hours 48 minutes.
Final E. M. F. per cell, 1.89 volts.
Average current per kilo., 1.74 amperes.
Energy given out per kilo., 8600 kilogrammeters.
Ampere-hours per kilo., 11.83.
Commercial efficiency, 80 per cent.

On the subject of the capacity of the accumulators and their efficiency, Mr. Gerard says that to store one electric horse-power-hour (270,000 kilogrammeters) it requires:

180 kilos. of Planté's accumulators.

60 kilos. of Faure's accumulators, with an efficiency of only 50 per cent., according to the experiments made at the Conservatoire des Arts et Métiers in Paris.

According to the above, $\frac{270000}{10700}$ or 25 kilos. of Julien's accumulators give one electrical horse-power hour with an efficiency of 80 per cent.

The following diagrams of discharge of a Julien battery (Figs. 5 and 6) composed of 29 cells of 88 pounds each, used by the Edison Company in Paris, show its efficiency and regularity of action charged at a rate of 15 ampere-hours per 2 pounds, or 660 ampere-hours in all, and discharged at the rate of 13½ amperes per 2 pounds, or 594 ampere-hours in all, with an efficiency in ampere-hours of 90 per cent. It will be perceived that the E. M. F. of the battery was 2 volts after twenty-two hours of continuous discharge:

Fig. 7 shows the fall in potential of a Julien battery, discharged to 0.7 volt (open circuit), beginning at 2.2 volts and maintaining an E. M. F. of 2 volts for 16½ hours, at the rate of 10 amperes, the remaining 27½ hours of the discharge showing a gradual fall to 0.7.

Fig. 8 shows the curve of the amperes in discharging the same battery, beginning at 10 amperes, and remaining constant for 22½ hours, the current falling gradually to 0 on the 44th hour.

The total weight, including retaining jar and acid, was about 30 pounds.

The ten negative plates contained in the above element were carefully weighed, and the amount of spongy lead estimated to be 4.5 pounds. As only half of the active matter in each of the two outside plates of an element is available, the actual amount of spongy lead concerned in the discharge was 10 per cent. less, or 4.05 pounds.

Now, 4.05 pounds of spongy lead will yield, theoretically, 486-

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volt ampere-hours, and the amount actually yielded in the experiment, as shown by the above diagrams, was 480-volt ampere-hours. This practical confirmation of the theoretical calculation is most gratifying.

From the above, it is quite evident that we have all the necessary data at hand to calculate the cost of furnishing light or power by accumulators, provided we know the cost of the batteries and their life, or, what is the same thing, their cost of maintenance.

It will be unnecessary for me to go into calculations relating to the horse-power required to move so many hundred tons of ore or

FIG. 5.

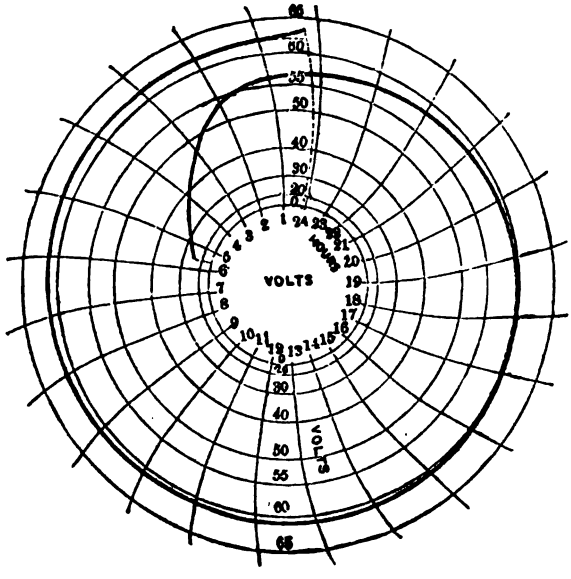

Diagram of Discharge of a Julien Battery (Fall in Potential).

coal per day so many miles with a given load and in a given time. This would vary in every individual case, and if I simply point out the cost of generating an electrical horse-power and of maintaining it with batteries, it will be an easy matter to apply it to any special case.

The principal reason why there has been so little progress in the introduction and use of storage-batteries is, in my opinion, not that they were not able to stand up under the work required of them, but that their cost was enormous. We have, therefore, attacked the problem entirely from a commercial point of view, assuming that we had a first-class battery to start with, or at least the best that has

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ever yet been made; and I think the result of our labors in this direction are of the most gratifying nature.

We received from abroad the battery, but the crudeness, and consequently the costliness, of the methods of manufacture imported with it rendered it hopeless to ever expect to accomplish anything on a commercial scale. In the short space of two years we have so changed the process that we are able to make eight times the number of batteries with the same number of men. With the force formerly required to make 100 cells per day we can now make 800. Here,

FIG. 6.

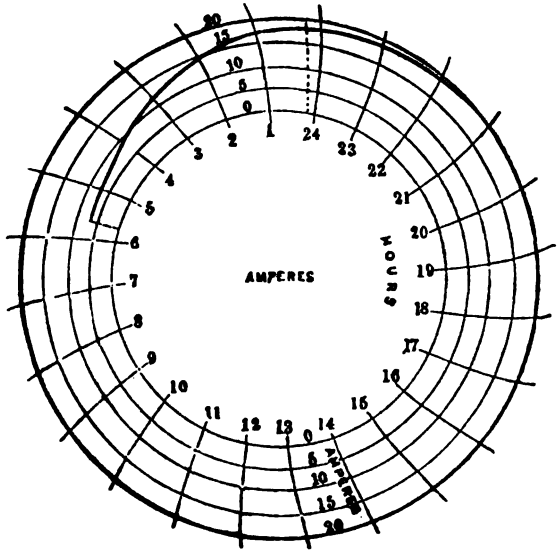

Diagram of Discharge of a Julien Battery (Fall in Current).

then, we are met with the crucial question: how much will it cost to introduce this system, and after its introduction how much will it cost to maintain it?

We have seen from the above that it requires 25 kilos. or about 55 pounds to generate 1 horse-power-hour; therefore, if the work to be done required 100 horse-power-hours, or 10 horse-power for ten hours, it would manifestly require 5500 pounds of battery. As each element of 19 "B" weighs about 25 pounds, this would mean 220 elements, and, at our present list-price of $8, would cost $1760. I have no hesitation in stating that we will be able to bring this price down one-half, owing to improved methods of manufacture.

The cost of maintenance per horse-power would be, allowing 2 cents per horse-power per hour, $2 per day or $600 per year of three

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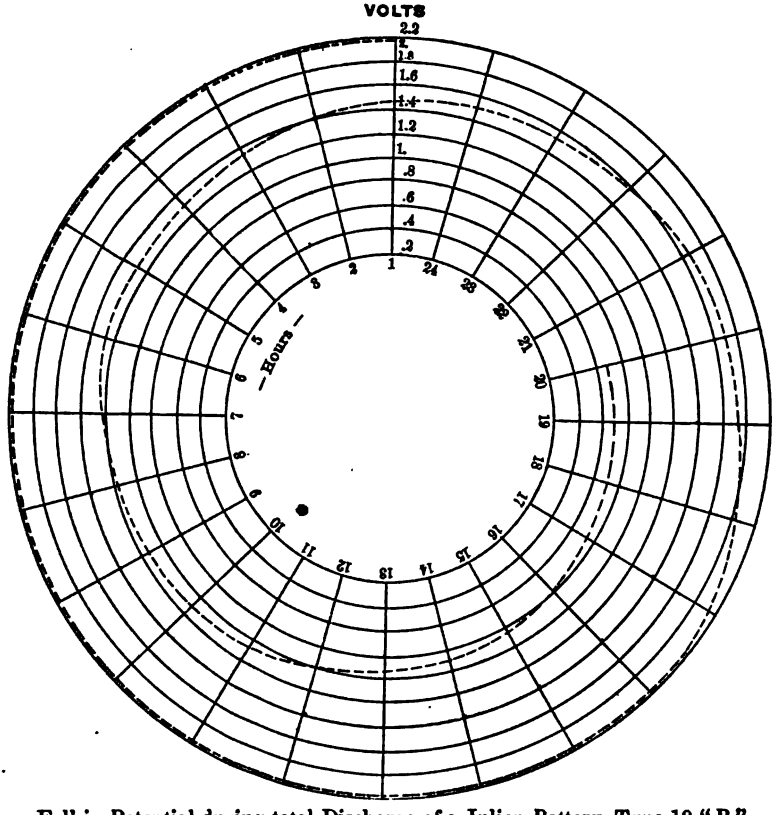
hundred days; and assuming that the batteries only lasted one year before the positive plates had to be removed, the cost of renewals would be about $400, or, say, as high as $600. This would make the cost of a horse-power 4 cents instead of 2 cents. I venture the assurance that the price of batteries will so steadily decline, when once they are introduced in sufficient numbers, that the increased ex-

FIG. 7.



Fall in Potential during total Discharge of a Julien Battery, Type 19 "B."

pense per horse-power-hour from using them will only be a fraction of a cent.

It may be of interest to some of our members who are engaged in mine-traction to know that the actual amount of power required to propel one of our 18-foot cars on the Fourth Avenue line in New York, from Eighty-fifth Street to the Post-Office and return, about 11½ miles, is 15 electrical horse-power-hours. At 2 cents, this is 30 cents, or $2\frac{5}{10}$ cents per car-mile.

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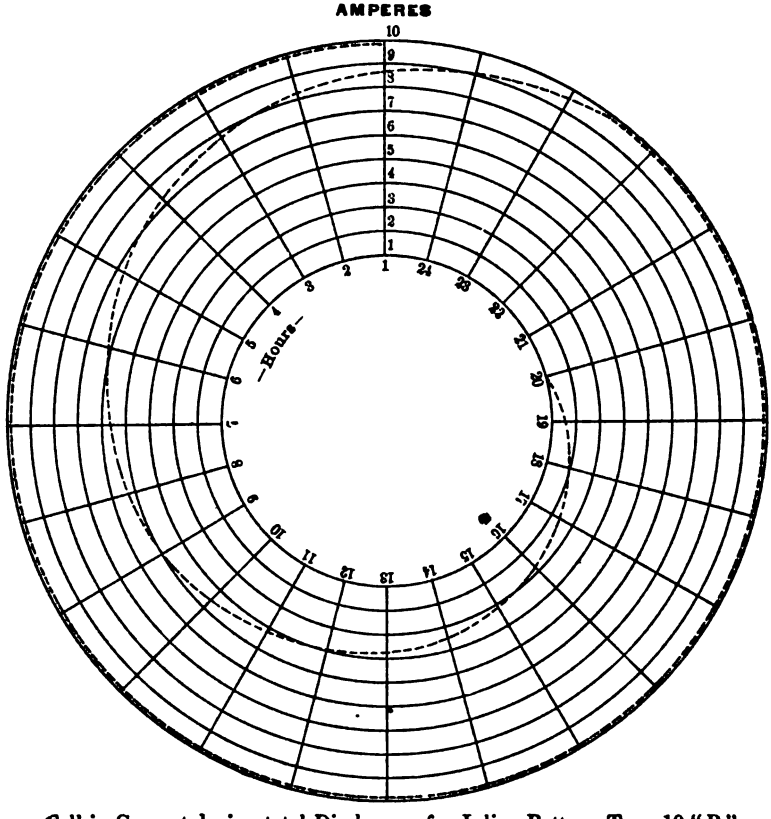
The cars on which this result was obtained have two trucks with eight wheels, and two 15-horse-power motors, one on each truck. The gearing from the motor-shaft to the counter-shaft is by gears *en echelon*. The power is transferred by means of chains to the sprocket-wheels on the axles. 144 cells are used, coupled in four groups, so that the E. M. F. at the motor terminals can be 75, 150, 225, or

FIG. 8.



Fall in Current during total Discharge of a Julien Battery, Type 19 "B."

300 volts. This allows four different speeds. After such a car has made a round-trip of 11½ miles, it requires, as I have said, 15 electrical horse-power-hours to replace the energy drawn from the batteries. We are able, however, to make two, or even three, round-trips, or about 36 miles in all, without changing the batteries. I am indebted for the above information to Mr. J. C. Chamberlain, Electrical Engineer and Superintendent of the Julien Electric Traction Company.

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NOTE ON GOLD-MINING AND MILLING IN KOREA.

BY WILLARD IDE PIERCE, NEW YORK CITY.

(Washington Meeting, February, 1890.)

THE native methods of mining and milling gold-ores in Korea may not present any specially new features, but are chiefly interesting as primitive practices, still followed at the present day.

In mining where the rock is hard, wood is piled up near the face of the tunnel or drift and then set on fire. The fire burns for twenty-four hours, after which, three or four days are given for the rocks to cool, before the miners go in and extract the loosened material. The only tools in use are a hammer, a steel gad about five inches long, and a flat spoon-shaped basket. The pay-streak or chute is closely followed, and in places the workings are very small and intricate. Rock not yielding more than $6 per ton is not extracted. The manner of opening a mine is to begin on the outcrop, where the rock pays, sink a shaft or run a tunnel on the vein, and follow the pay-streak and more easily worked ore. When this is exhausted, or the influx of water interferes, the opening is abandoned and operations are begun again at a new spot on the outcrop. The place usually selected is as high up on the mountain as possible, and in no instance is advantage taken of the topography by running a cross-tunnel to tap the vein.

The ore is brought in baskets to the surface, and there placed in sacks, holding about 150 lbs. each, and taken to be crushed. The rock is first crushed between two stones, as shown in Fig. 1. The upper stone is oval and rounded on the bottom, so that it can be readily rocked on the under stone. Two pieces of wood, used for handles, are placed, one on each side, in grooves in the upper stone, and held in position by means of straw rope. The lower stone is not smooth on top, but exhibits three elevations running lengthwise, with shallow depressions between them. The middle elevation acts as a fulcrum on which the upper stone may be rocked. From two to ten men, according to the weight and size of the stone used, are employed in rocking. With four men, from 300 to 350 lbs. of rock can be crushed per day, the stone making 50 complete oscillations per minute. The men squat on each side of the stone and move it to and fro by means of the wooden handles. The ore is fed, in pieces up to the size of an egg or even larger, by one of the men. During the crushing one man on each side, as the stone rocks from

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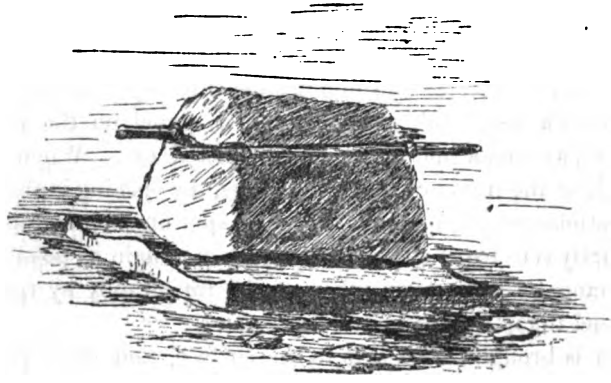
him, uses a stick as a scraper to stir the crushed rock. When the ore is fine enough, it is swept out on a piece of cloth, placed at one side of the stone, the rocking being continued all the time. After a certain amount has been crushed, it is usually put through a horse-hair sieve, and the coarser portion is treated again. An upper stone, intended for four men, measured 23 inches in length, 20 inches in width at the base and 15 inches in height. Crushing is usually performed in sheds, to prevent the wind from blowing any of the fine stuff away.

The crushed ore is taken to men who rub it between stones until it is very fine; the rubbing is generally done wet. The powdered ore is received by other workmen, who pan it and save the free gold. The pan is of one piece of wood, 20 inches in diameter and 5 inches high; and the men who use this instrument are the most expert

FIG. 1.



Korean Gold-Mill.

panners I have ever seen. The pan is not held in the hands, but floats on the surface of the water, one side only being raised or lowered to agitate the contents. As the pulp is very fine, and the gold in the same condition, the top of the water in the pan is frequently sprayed, so as to settle any gold that may be floating. The first tailings are washed off directly from the right-hand side of the pan. After this the sulphurets and gold are collected on the right-hand side of the pan, and the tailings on top of the sulphurets are, by a few skillful movements, washed over to the left-hand side. The pan is then half revolved, and the tailings are washed off from the right-hand side. The same operation is repeated after the tailings have all been removed with the sulphurets and gold, the former being saved and again rubbed. No quicksilver is used; but I do not think that there is any loss in free gold.

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BIOGRAPHICAL NOTICE OF CHARLES A. ASHBURNER.

BY J. P. LESLEY, PHILADELPHIA, PA.

(Washington Meeting, February, 1890.)

THE old do not love to see the young pass away from the light of the sun before them. Fathers would fain keep their sons by their side to the end of life; but the old Greeks, who loved the old gods, were wont to moderate their grief with the sweet superstition that only those whom their gods especially loved died early. The Christian church found consolation in that superstition applied in a new form to its new sorrows, and paid its most enthusiastic devotion to the memories of its young and beautiful martyrs. The natural science of our century is robbing us fast of this and all other superstitions, sweet as well as bitter, and leaving us for consolation to the teaching—colder, yet kinder—of personal fortitude and that optimism which intelligently translates the Cosmos of Humboldt back into its old name of the Harmonia of Pythagoras. The Homeric Kataclothés, the three fates, are dead and gone for us, with that old world which comprehended none of the laws of cause and effect, and sorrowed for those who were cut off from the land of the living without hope of more than a shadowy existence beyond the river of death. All the more the ancients cherished the memory of their dead and lavished their choicest art upon their monuments. We moderns have lost the monumental arts, but we better keep the monuments which our dead leave behind them. No one of us who has done good work can fail to be remembered; and in an Institute like this, which keeps the press at work, an immortality in the memory of men is more possible, more certain, for every one of its members than the greatest heroes of antiquity could anticipate for themselves. In old apocalyptic times the works of men followed them through death to the throne of God to be their advocates in judgment, but in these modern times our works remain this side the grave, to follow the name of the departed one as it takes its course along the history of his peculiar art or science advancing slowly to perfection. We write the epitaph, not upon the mouldering stone of a tomb, but on the pages

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of a book which shall stand in all the libraries of Christendom. Let it be written lovingly.

What, then, are the monuments which our Charles A. Ashburner has left behind him? He was your fellow-member, and you grieve for his sudden loss; he was one of my children, and I grieve still more. To me his death is one of the irreparable losses. He was one of my college boys. I taught him the elements and principles of geology in the Towne School of the University of Pennsylvania from 1872 to 1875. He was one of the quickest of learners, and took to physical science like a duck to water. He had a genuine genius for appreciating form and structure, and was one of the few who, at the outset of a scientific career, comprehend the uses of accuracy. Many never learn them; he was accurate by nature. Everything of the nature of true proportion appealed to him with the certainty of a response in the shape of some additional striving after absolute precision of statement or presentment. He was a born artist, seeing what he drew and drawing what he saw. The love of exact scientific truthfulness, however, in his case never hardened, as it does in so many other cases, into a pedantic formalism. He was full of inventiveness. His imagination was fertile in new inventions for discovering and portraying the exact proportions and relations of things—the objects of inquiry. He was a zealot in science. He might have said to any one, or to his own soul, with safety, "The zeal for exact truth has eaten me up." In fact, we owe this zeal the heavy grudge that in the end—nay, not half-way to the proper end—it killed him. He never spared himself, or any price, to become perfectly sure of facts. For many of his facts he had to pay a high price; but the actual facts he would have. No half-facts for him. I have not encountered a more real and typical man of science—born for true science. Consequently he was a discoverer, a natural leader of men in exploration.

His first work was topographical. I commissioned him and his classmate, Charles E. Billin, in 1875, to aid Mr. John H. Dewees, Assistant Geologist in charge of the survey of the fossil ore-belt of the Juniata Valley. They soon learned to carry on their geological as well as their topographical work independently of Mr. Dewees. A very perfect contour-line map of the south flank of Jack's Mountain, and of the small valleys and ridges in front of it, was made by them in common, and is one of the most satisfactory products of the State survey.

Mr. Ashburner wrote the report on the Aughwick Valley and

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East Broad Top Coal-basin, published, with Mr. Dewees's report on the fossil-ore mines, as "Report F," in 1878. His discussion of the Three Springs fault showed his more than ordinary geological ability, and was a prediction of his future eminence. But the many precisely-constructed sections across that belt of the State, published in Report F, proved that he combined the qualities of geologist and artist in the highest degree.

In 1876 I commissioned Mr. Ashburner to survey one of the most important districts of the State, and one of the most difficult—the counties of McKean, Elk, Cameron and Forest, containing the Bradford oil-district, then becoming famous for its productiveness. His survey of McKean was commenced in July of that year, and lasted two years; but his report on it (R) was not published until 1880. The district being traversed by gentle anticlinal waves, which but slightly modify the essential horizontality of the Lower Carboniferous and Devonian measures, and being occupied largely at the surface by the Conglomerate No. XII., which had not then been subdivided properly in northern Pennsylvania, although its subdivisions had been made out by I. C. White in the western counties, Ashburner instinctively felt that his success would depend on a good topographical map as the first step of the survey; and he made one of the best, contouring it with the eye of an artist who knew the geological significance of every feature of every curve. When a model in relief on an equal vertical and horizontal scale was made from this map, and the colored belts of the outcrops were laid upon it, no more perfect exhibition of the geology of an extensive area could be imagined. But his study of the underground by surface-sections and well-borings was quite as excellent, and quite as well expressed to view. The generalizations which he deduced from it bore the most important fruit, fixed the limits of the oil, and placed the calculation of boring-depth in that district on a sure scientific basis; one of his most striking discoveries being the rapid increase in thickness southeastward of formation No. X., from 250 feet to 750 feet, which explained the failure of many borings to reach the deeper oil-horizon in Elk and Cameron counties. In 1878 Mr. Arthur W. Sheafer was commissioned to assist him in completing the survey of the four counties; and the second volume (R R), relating to Elk, Cameron and Forest, was published in 1885, having been long delayed by his work in eastern Pennsylvania.

When, in 1880, the time came for organizing the survey of the anthracite-region as a special and most important part of the survey

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the State, I selected without hesitation Mr. Ashburner to plan, organize and execute it. I have no intention of describing this *chef d'œuvre* of geology as an applied science, now famous at home and abroad; for its numerous sheets of mine-maps, columnar sections and cross-sections are probably in the hands of most of the members of the Institute. I wish only to lay the fame of this splendid achievement as a green wreath on the tomb of our fellow-member. He knew exactly what was to be done and did it. He selected his assistants, taught them and worked with them, inspired them with his own zeal, and lifted their work to the standard level of his own, and kept it there. He encountered indescribable obstacles of social, mercantile and professional kinds, and overcame them with admirable tact and good judgment. He entirely conquered the rooted prejudice of practical miners and local engineers against scientific geologists, until it became evident to all that the State survey knew what it was about, was doing a special and specific business, and not only could teach the oldest and most intelligent operators something they did not know, and unaided could not learn, but would place upon their office-tables what they would soon come to consulting every day, and would not part with at any price. Above all, he was wise enough to inspire everybody in the anthracite region with entire confidence in his honesty, in his truthfulness, in the exact meaning of what he promised to do and not to do, and in his certain performance of such promises. His sense of private and professional honor was so keen and so subtle that it led him triumphantly through a perfect labyrinth of suspicion, fear and dislike, engendered against him and his survey by contending colliery-interests and competing official interests. He insured accuracy for his own work, and confidence in it at the same time, by the original device of a submission of every proof received from the artists of every sheet in its stages of publication, to the superintendents and engineers of the colliery companies, to be criticized and corrected as they pleased. But these returned proofs were then subjected to re-examination by him and his assistants, to test the value of such corrections, over which conferences were held, and debates, until they were accepted or rejected. In many cases the companies themselves saw the necessity for new and, at first, strange work, and ordered it done by their own engineers. In a word, not to be tedious, the survey produced a change of professional sentiment in the whole region, of a nature which the members of this Institute can well understand without my describing it. This was Ashburner's

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doing. Of course his whole corps of assistants ably supported and seconded him; fell into his ways; helped to make his system successful; became themselves able geologists; and are now among the most trustworthy and reputable. His will was strong, but his heart was warm; and, while he permitted no disobedience of orders, I never knew a man more generous and faithful to those above and those below him. Such a man will make enemies; but he never showed the least rancor towards them. I have said he was a typical man of science; I can safely add that he was a true Christian gentleman, with a heart overflowing with affection to his fellow-men. Ambitious? yes, very ambitious, but only of power which he never abused; wealth which was not for himself; fame, but of the noblest kind.

Mr. Ashburner conducted the anthracite survey from 1880 to 1887, and was succeeded by his accomplished assistant, Mr Frank Hill, who completed the survey June 1, 1889; when by Act of Assembly the work of the Geological Survey of Pennsylvania ceased and the whole corps was disbanded. Mr. Ashburner's first work was a thorough survey of the Panther Creek, or eastern division of the southern anthracite field, between Mauch Chunk and Tamaqua, his report on which (AA) was published in 1883. In subsequent years he had separate field parties, working simultaneously, in the Eastern middle, Western middle, and Northern fields, with offices at Pottsville, Hazleton, and Wilkesbarre, his headquarters being at Philadelphia.

In the fall of 1886 he resigned his commission (with the understanding, however, that he would give half his time still until the following summer) to accept business relations, as a scientific expert, with Mr. Westinghouse at Pittsburgh. Since then, and up to the time of his death, he travelled widely in the United States and Canada to examine especially new oil- and gas-fields, and latterly proposed plants for mining the precious metals. It was on his second return from Arizona in December last, worn out with exertion, exposure and the responsibilities of his office, that he fell ill and died at Pittsburgh, leaving an amiable wife and two young children and innumerable friends to mourn his loss.

He was a member of the American Philosophical Society from 1880, in the *Proceedings* of which will be found the following papers, which he read at the meetings of the Society between 1881 and 1889:

On Kintz's Fire-Damp Indicator, xxi., p. 283.

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Notes on the Natural Bridge of Virginia, xxi., p. 699.

Remarks on the Recent Publications of the Second Geological Survey of Pennsylvania, xxii., p. 86.

Mr. Ashburner joined the American Institute of Mining Engineers in 1875, served as Manager in 1885, 1886 and 1887, and contributed to the *Transactions* the following papers:

	Transactions.
1. The Bradford Oil-District of Pennsylvania,	vii., 316
2. The Brazos Coal-Field, Texas,	ix., 495
3. New Method of Mapping the Anthracite Coal-Fields of Pennsylvania,	ix., 506
4. The Flannery Boiler-Setting for the Prevention of Smoke,	x., 212
5. The Anthracite Coal-Beds of Pennsylvania,	xi., 136
6. The Product and Exhaustion of the Oil-Regions of Pennsylvania and New York,	xiv., 419
7. The Geology of Natural Gas,	xiv., 428
8. The Classification and Constitution of Pennsylvania Anthracites,	xiv., 706
9. The Geologic Distribution of Natural Gas in the United States,	xv., 505
10. The Geologic Relations of the Nanticoke Disaster,	xv., 629
11. Coal-Production in Utah, 1886,	xvi., 356
12. Petroleum and Natural Gas in New York State,	xvi., 906
13. The Development and Statistics of the Alabama Coal-Fields for 1887,	xvii., 206
14. The Geology of Buffalo as Related to Natural-Gas Explorations along the Niagara River,	xvii., 398
15. The Coal-Trade and Miners' Wages in the United States for the year 1888,	xviii., 122
16. Natural-Gas Explorations in the Eastern Ontario Peninsula,	xviii., 290

He also read at the Ottawa meeting, in October last, a biographical notice of Capt. W. R. Jones, of Pittsburgh, whose recent and distressing death by accident we were all at that time mourning. The manuscript of his notice of Capt. Jones he retained for final perfecting. It must now be finished by another hand, and the same last service must be done for him who undertook it for his friend.

THE FORM OF CRATER PRODUCED BY EXPLODING GUNPOWDER IN A HOMOGENEOUS SOLID.

BY FRANK FIRMSTONE, EASTON, PA.

(Ottawa Meeting, October, 1889. Substituted for the briefer note on the same subject, presented at the meeting and subsequently distributed in preliminary pamphlet edition.)

Prof. H. Hoefer's theory of blasting was published in the *Oesterr. Zeitschrift*, xxviii., 1880, and xxix., 1881; and Part III. in Nos. 15, 16 and 17, xxx., 1882, and subsequent numbers of that

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year. My attention was called to the matter by Major C. W. Raymond, Corps of Engineers U. S. Army, to whom I am indebted for a copy of his translation of Parts I. and II. Referring to the original memoir for the details and the analysis employed, I may sum up the essential points as follows:

It is assumed that the crater formed in a very large mass of a moderately resisting isotropic substance, having a plane as the free surface, will be essentially a right cone, with the apex at the center of the charge, and that the forces from an explosion, acting radially at any two points in the mass, will be to one another inversely as the squares of the distances from the center of the charge; also that the limit of the crater, when one is formed, will be defined by those points on the free surface at which the component of the radial force normal to the free surface is just sufficient to break loose and displace a particle of the material in which the blast is made.

It is also tacitly assumed that the charge is deposited in a spherical case of small radius compared with the line of least resistance.*

As a consequence of these premises it is shown that the maximum volume of the crater with a given charge corresponds to that line of least resistance which makes the angle at the base of the crater 48° 11′ 23″, and that no crater will be formed if the line of least resistance be made 1.554 times as long as that which corresponds to this normal (maximum) crater. The author points out that his theory confirms certain practical rules of long standing in military mining, and also shows a close agreement between calculated and measured quantities, when it is applied to certain mining experiments made by the Austrian engineers at Olmütz.

A year or more ago I made a series of experiments to test, as far as was practicable on a small scale, the assumption of this theory as to the form of the crater.

My experiments were made by pouring liquid plaster-of-Paris around a charge of gunpowder enclosed in a water-tight cartridge, and firing by electricity after the plaster had hardened. Gunpowder was used because, with any explosive requiring a detonating primer, the explosive effect of the primer would require a large and altogether uncertain correction for different weights of charge.

The primers employed were simply the platinum bridges used for the Laflin and Rand low-tension fuses, having about two feet of insulated copper leading-wires. They were securely lashed to small

* See Part III., *Die gekuppelten Minen.*

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slips of wood, about 3 inches long, to prevent the platinum bridges from being broken by any movement of the wires in making the cartridges.

The powder used was all taken from the same 6-pound case of sporting powder, the whole of the contents being poured into a large basin, thoroughly mixed and exactly weighed in quantities of 1, $2\frac{1}{2}$, 5 and 10 grammes, which were kept in well-corked vials until used.

The cartridges were made of thin writing-paper, folded like a plaited filter, and were worked into a roughly spherical shape around the powder and primer, and well choked with thread around the stick to which the primer was lashed, care being taken that no powder was left in the folds of the paper. They were rendered water-proof by repeated dipping into melted paraffin, which also filled up the irregularities of the paper and enabled me to mould the outside very nearly into spherical form. They were then tested with a galvanometer to see that the platinum wires had not been broken—an accident which in fact did happen several times.

The blocks of plaster were made in wooden moulds, having bottom, movable sides and ends of boards. A hole was bored in the center of the bottom-board, and a piece of wood about $\frac{3}{8}$-inch in diameter was stuck upright in it. To this the cartridge had been previously fastened by a piece of No. 12 copper wire laid over the joint between the end of the upright stick and the slip of wood projecting from the cartridge, and well secured with thread. The length of the piece of wood stuck in the bottom-board was adjusted to bring the center of the cartridge to an exact height above the bottom. The cartridge was centered in the mould by measuring from the sides, and bending the copper wire until it was in the proper position.

The plaster was mixed in the proportion of 4 parts, by weight, of plaster to 3 parts of water, and the quantity for each block was weighed out beforehand in two or three lots, according to the size of the block, and mixed and poured into the mould as rapidly as possible. From the first lot mixed for each block four standard briquette-moulds were filled, and subsequently broken in a Riehlè cement-tester, to detect variations in the strength of the blocks. In most cases three blocks, intended to have the same dimensions and to be used with the same charge, were made at one time, the sides of the moulds being removed as soon as the plaster had fairly set. The briquettes were trimmed and placed on top of the blocks, so that they might be exposed as far as possible to the same hygro-

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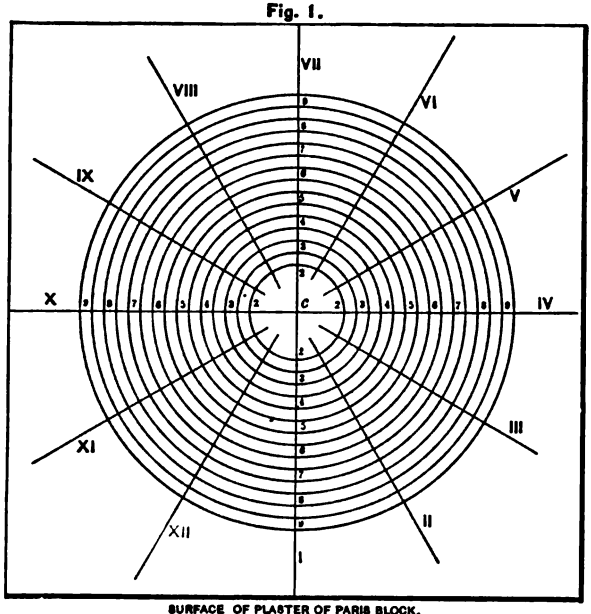
metrical conditions, and were broken as soon as possible after firing. The blocks stood from 24 to 48 hours before the charges were exploded. Before firing lines were drawn on the top of the block to facilitate the examination of the crater (Fig. 1).

Fig. 1.



SURFACE OF PLASTER OF PARIS BLOCK, AS PREPARED FOR MEASURING RESULTS OF EXPERIMENTS IN BLASTING.

After firing the crater was carefully cleared of any loose material which had fallen back into it; the fragments blown out were examined, and the volume of the crater was measured by filling it with No. 4 shot, one pound of which was found by careful trial to correspond to 3.96 cubic inches (apparent specific gravity 7). The outline of the crater was then traced in full size on tracing-cloth, and the shot were removed.

The block was next sawed in two with a cross-cut saw, preference being given to the line apparently most nearly free from accidental irregularities; and the vertical section thus obtained was traced on cloth, the center, top and bottom of the blocks being carefully marked. From these sections the line of least resistance was determined for each case, because blocks from a given weight of plaster were found to vary considerably in thickness.

The areas of the tops of the craters were measured from the tra-

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TABLE OF EXPERIMENTS IN BLASTING.

Number of Shot.	Charge in Grammes.	T Tensile Strength of Plaster, lbs. per sq. in.	W Line of Least Resistance. Inches.	K Diameter of Cartridge. Inches.	K Volume of Crater. Cubic Inches.	K Area of Top of Crater. Square Inches.	K Diameter of Circle of Equal Area. Inches.	Size of Block. Inches.	Thickness of Block. Inches.	Remarks. (The Roman Numerals refer to Fig. 1, page 373.)
16 a	2½	120	3″	¾	105.9	113.73	12.05	24	7	
b	2½	110	3⅛	¾	100.9	109.93	11.85	"	7	
c	2½	107	3	¾	50.2	64.31	9.05	"	7	
17 a	2½	119	4	¾	.....	.....	.....	24	9	No report. Block split in two.
b	2½	112	4¼	¾	258.3	173.94	.....	"	9¼	No report. Oblong piece blown out in direction of diagonal.
c	2½	116	4	¾	.....	.....	.....	"	9¼	No report. Split in two nearly on line X—IV. No crater.
18 a	2½	127	4	⅝	.....	.....	.....	24	9½	Split in two. No crater.
b	2½	123	4⅛	⅝	204.1	172.14	14.8	"	9½	Regular crater. No crack in block.
c	2½	121	4⅛	⅝	228.4	203.75	16.15	"	9½	Good crater. No crack in block.
19 a	2½	116	4⅛	⅝	358.9	303.00	19.65	24	10¾	
b	2½	121	4⅛	⅝	.....	.....	.....	"	10¾	No distinct crater. Three corners and most of sides IV, VII and X blown away.
c	2½	116	4⅛	⅝	344.8	245.6	17.65	"	10¾	
24 a	2½	128	4	¾	161.7	142.6	13.5	24	10	Sharp report. Very regular crater.
b	2½	110	4	¾	.....	.....	.....	"	10⅛	Moderate report. Split in two parallel to line II—VIII and 2″ from it.
c	2½	106	3⅛	¾	174	222.82	16.85	"	10	First cartridge failed and bored out from back. New cartridge put in and tamped with plaster. Very regular crater. One large piece blown out in center.
25 a	2½	91	5	¾	.....	.....	.....	24	12	Moderate Report. Split in two on line passing from VII, 6″ to left of I.
b	2½	101	5	¾	.....	.....	.....	"	12	Moderate report. Corners (V—VI) (VIII—IX) (XI—XII), all of sides VII and X and most of I and IV blown away.
c	2½	99	5	¾	.....	.....	.....	"	12	Moderate report. Corners all stood. Piece 14″ long, 1⅝ deep in center, blown off side X.
35 a	2½	93	2½	¾	40.3	62.64	8.9	30	9	Intended for 11″ thick, but mould leaked. Moderate report. Small, regular crater.
b	2½	90	4	¾	222.0	206.62	16.25	"	10½	Small report. 4 large, several small pieces blown out. Crater rather irregular.
c	2½	89	5½	¾	.....	.....	.....	"	11½	Very small report. Block split in two, nearly on line X—IV.
39 a	2½	93	5	¾	.....	.....	.....	30	11	No report. No visible effect, except hair crack on bottom about 12″ long. No powder stain.
b	2½	98	5	¾	.....	.....	.....	"	11	No report. No visible effect, except irregular crack on top with slight powder stain.
c	2½	97	5	¾	368.3	285.8	19.10	"	11	Slight report. Fairly regular crater. 4 large and several small pieces blown out.
40 a	2½	97	5	¾	.....	.....	.....	30	11½	Very small report. No crater. Top slightly raised and cracked on parallel lines about 6″ apart. No crack on sides.
b	2½	101	5	¾	.....	.....	.....	"	11	Very slight report. Top raised and cracks on two lines parallel to sides I and VII, about 6″ from them. Cracks
		88	5	¾	.....	.....	.....	"	.....	on sides V and VII 1⅛″ below top.
c	2½	5	1″	¾	142	142.7	13.5	24	7¼	Sharp report. Cartridge displaced in filling mould and crater formed both top and bottom. Top crater.
a′	5	84	3½	1″	107	154.9	14.05	"	"	Bottom crater. Values of W not certain.

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b	5	93	3 3/8	1″	76.7	91.43	10.85	"	7 1/4	Sharp report. Crater regular. Pieces blown out all small.
c	5	95	3 1/8	1″	77.7	90.87	10.75	"	7 1/4	Sharp report. Crater regular. Pieces blown out all very small.
29 a	5	108	4 3/32	1″	232.7	206.02	16.20	24	10 1/2	Small report. Crater regular. Pieces blown out moderate in size.
b	5	115	4	1″	203.9	184.09	15.35	"	10 1/2	Moderate report. Crater regular. Pieces blown out moderate in size.
c	5	104	4 3/16	1″	215.1	191.25	15.65	"	10 1/4	Moderate report. Crater regular. Pieces blown out smaller than in 29 a and b.
32 a	5	107	5 1/32	1″	.....	.....	.....	24	12″	Small report. Corner (XI—XII) blown away and most of side X.
b	5	102	5	1″	.....	.....	.....	"	12 1/4	Moderate report. All corners and sides blown off. Top broken into 4 large and several small pieces.
c	5	113	5 1/16	1″	.....	.....	.....	"	12 1/4	All corners except (XI—XII) blown away. Pieces of top large.
36 a	5	88	5	1″	338	248.7	17.80	30	11	Moderate report. Crater regular. 3 large pieces and several small ones blown out.
b	5	86	4 13/16	1″	282	217.2	16.65	"	11	Small report. Crater regular. Pieces smaller than 36 a.
c	5	97	4 13/16	1″	329	252.3	17.95	"	11	Moderate report. Crater regular. Pieces much broken up.
27 a	7 1/2	90	3 3/8 (?)	1 1/8	171.6	176.21	15.00	24	7 1/4	Sharp report. Large crater top and bottom. Cartridge displaced in filling mould.
a′	7 1/2	90	3 1/2 (?)	1 1/8	122.2	166.94	14.6	"	"	Bottom crater, N. B. block was also split in two on an irregular line. Value of W uncertain.
b	7 1/2	91	3	1 1/8	65.3	71.38	9.53	"	7 1/16	Sharp report. Crater regular. Pieces thrown out small.
c	7 1/2	92	3 3/16	1 1/8	64.4	67.72	9.29	"	7 1/8	Sharp report. Crater regular. Pieces thrown out small.
30 a	7 1/2	99	4 1/16	1 1/8	110.9	109.51	11.80	24	10 1/4	Moderate report. Crater very regular. Pieces blown out small.
b	7 1/2	101	4	1 1/8	204.9	162.73	14.4	"	10	Moderate report. Crater not so regular as 30 a. No piece over 2″ cube blown out.
c	7 1/2	112	4 1/16	1 1/8	174.2	166.06	14.55	"	10 1/4	Moderate report. Crater regular. Pieces rather larger than 30 a and b.
33 a	7 1/2	112	5	1 1/8	.....	.....	.....	24	12 1/8	Moderate report. All corners except (V—VI) blown away in large pieces. Pieces from center large.
b	7 1/2	105	5	1 1/8	.....	.....	.....	"	12 1/8	Moderate report. Corners (XI—XII) (II—III) stood. Block split in two on line IV—X.
c	7 1/2	102	5	1 1/8	233.2	197.0	15.8	"	12 1/8	Moderate report. Block split in four on lines (I—VII) (IV—X). Also a very regular crater when they were forced together again.
37 a	7 1/2	96	4 3/4	1 1/8	252.2	207.21	16.25	30	11″	Moderate report. Crater regular. Pieces all small.
b	7 1/2	84	4 1/2	1 1/8	273.5	193.47	15.70	"	11 1/16	Moderate report. Crater regular. Pieces small.
c	7 1/2	91	4 1/16	1 1/8	355.5	283.18	19.00	"	11 1/16	Moderate report. Crater regular. Pieces larger than 37 b or c. Block split in four nearly on lines (I—VI) (IV—X).
28 a	10	116	3 1/4	1 3/16	72.3	77.25	9.92	24	7 1/4	Sharp report. Crater regular. Pieces blown out small.
b	10	122	3 1/8	1 3/16	75.2	78.15	9.98	"	7 1/4	Sharp report. Crater regular. Part blown out much broken up.
c	10	111	3 1/4	1 3/16	101.2	98.14	11.20	"	7 1/4	Sharp report. Crater regular. Part blown out much broken up.
31 a	10	104	4 1/16	1 3/16	155.9	144.23	13.55	24	10 1/8	Moderate report. Crater regular. No pieces exceeding 3″ x 2″ x 1½″ among those blown out.
b	10	109	4 1/16	1 3/16	135.6	121.36	12.45	"	10 1/8	Moderate report. Crater fairly regular. Pieces blown out a trifle larger than 31 a.
c	10	116	4	1 3/16	131.7	123.22	12.55	"	10 1/8	Moderate report. Crater regular. Pieces blown out small.
34 a	10	107	5 1/16	1 3/16	.....	.....	.....	24	12	Moderate report. No crater. All corners blown away and block split on line (X—IV).
b	10	109	5 1/8	1 3/16	235.9	189.75	15.5	"	12 1/8	Sharp report. Crater regular. Pieces blown out small.
c	10	104	5 1/8	1 3/16	.....	.....	.....	"	12 1/8	Moderate report. Sides (VII and X) blown away. Block split on line X—IV.
38 a	10	83	4 17/32	1 3/16	203.5	164.20	14.45	30	11	Moderate report. Crater regular. Pieces all very small.
b	10	87	5	1 3/16	382.9	283.53	19.00	"	11 1/4	Moderate report. Crater regular, much larger than 38 a, and pieces blown out larger.
c	10	100	4 31/32	1 3/16	251.2	204.19	16.15	"	11	Rather sharp report. Crater regular. Pieces blown out medium in size.
21 a	1	111	3 1/4	13/32	.....	.....	.....	24	7 1/4	No report. Gas escaped along stick supporting cartridge.
b	1	109	3 1/8	13/32	100	176.6	15.00	"	7 3/8	Slight report. Cartridge displaced in filling. Very regular crater formed on bottom, the plaster from which came out in a single piece.
c	1	111	3 1/4	13/32	.....	.....	.....	"	7 1/4	No report. Gas escaped along stick.
22 a	1	104	3 3/4	13/32	222.7	215.1	.....	24	7 1/4	Slight report. Cartridge displaced. A large piece with oval base easily detatched from bottom.
b	1	107	3 1/16	13/32	58.9	84.88	10.4	"	7 1/4	Slight report. Small, very regular crater formed.
23 a	1	114	3 1/8	5/32	.....	.....	.....	24	7 1/4	No report. Small crack visible in bottom.
b	1	115	3 1/8	5/32	.....	.....	.....	"	7 1/8	No report. Small jet of smoke escaped from a very fine crack in top.
c	1	116	3 1/8	5/32	.....	.....	.....	"	7 1/8	No report. No visible effect. Gas escaped through fine crack below. [resting on its side.
41 a	7 1/2	.....	3 3/16	1 1/4	49.87	63.05	8.96	24	7 1/2	Portland cement. First cartridge failed. Drilled out from back, reloaded and fired with the block
b	7 1/2	113	4 1/8	1 1/4	142.20	134.08	13.05	"	8 1/8	Portland cement. Loud report. Fairly regular crater. Pieces small. Block split into three.
c	7 1/2	74	3	1 1/16	54.31	89.98	10.70	"	7	Portland cement. Sharp report. Fairly regular crater. Pieces blown out small.

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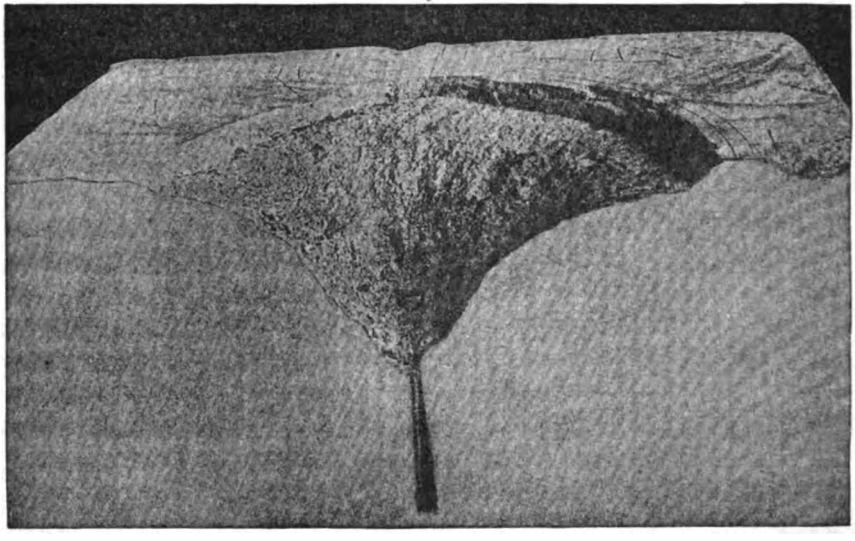
cings by a planimeter, and, with the diameters of circles of equal area, are entered in the table on pages 374 and 375.

In this way trials were made with blocks 15, 24 and 30 inches square, and charges of 1, $2\frac{1}{2}$, 5, $7\frac{1}{2}$ and 10 grammes of powder, and with lines of least resistance ranging from $2\frac{1}{16}$ths to about 5 inches.

The vertical sections in all cases, as shown in Figs. 2 to 12 inclusive, and in Fig. 20, which is a reproduction by the Ives process from a photograph, were strikingly similar, and departed widely from a cone.

To determine whether this shape was due to some special properties of the plaster-of-Paris, trials were made with Portland cement, which gave similar figures, but not quite so regular.* Moreover, I

Fig. 20.



View of crater as shown after sawing block in two.

have once observed similar craters in removing with dynamite an old blast-furnace bottom, which (originally of coarse-grained sandstone) had been converted into a mass free from joints by long heating and the infiltration of cinder.

That the shape was not due to systematic variation in the strength

* The cement being too hard to saw, a cast of the crater was taken in plaster, and this cast sawed in two to get the section. As it was very difficult to lay out lines on such an irregular body, it is probable that the section did not pass exactly through the center of the charge.

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of the blocks from the upper surface downwards was proved by breaking test-pieces, sawed from blocks made for the purpose, and also by putting the briquette-moulds into the 15-inch block-moulds, one on top of the other, separating them by slips of wood, and filling the mould to just cover the upper briquette-mould. After 24 hours the briquette-moulds were cut out, and the briquettes broken. Four trials in this way on blocks 11 inches thick (corresponding to the thickest blocks used in the experiment) were entirely conclusive on this point. An incidental proof is also furnished by shots 21b and 27a, in both of which the cartridge was displaced in filling the mould, resulting in the production of regular craters at both top and bottom (Fig. 13).

When the block was too small in proportion to the charge and the line of least resistance, all or nearly all the upper surface was commonly loosened, the fracture extending down the sides of the block, especially at the corners, in a remarkable manner. This was repeated several times with surprising regularity (shots 19*b*, 25*b*, 33*a*, 33*b*, 34*c*; Figs. 14, 15, 16, 17, 18, 19). In some cases (shots 25*a*, 35*c*, 39*a*, 39*b*) the block was split into two or more pieces without the formation of any crater, and in one case (shot 33*c*) a regular crater was formed, and at the same time the block was split into four pieces.

A good deal of labor was expended in searching for empirical formulas, whereby the observed values of the craters could be reduced to a common value of T; but the results were so unsatisfactory, on account of the small number and excessive irregularity of the data, that I do not think it useful to include them here.

In spite of this I deem it safe to conclude that the volume of the crater increases continuously with W until a maximum is reached, from which it falls abruptly to zero. This limit was at about W = 3 inches for the 1-gramme charges, and between W = 4½ inches and W = 5 inches for the 2-gramme charges.

I could not go far enough to find the limits for heavier charges, which would have required blocks 3 or 4 feet square at least, and much larger than I could make with the means at my disposal.

With the heavier charges there is a very considerable enlargement close to the charge; and in all, the crater plainly consists of two parts: an upper, spreading part, which increases with W, and nearly as W^3 for a given charge (if anything, rather faster than W^3, near the extreme value of W), and the enlarged part next the charge ("sphere of comminution").

The upper, spreading part, plainly decreases slightly (for a given

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value of W) as the charge increases, although, on account of the enlargement around the charge, small values of W may give larger craters with heavy charges than with light.

The influence of the tensile strength of the plaster cannot be so clearly made out. In the $2\frac{1}{2}$- and 5-gramme series it is plain that the crater increases with an increase in T, but this is apparently reversed in the $7\frac{1}{2}$- and 10-gramme series. When the variation of T was extreme it was evident that the effect of the powder was greater with a large value of T.

Shots 9*a*, 9*b*, and 9*c* (not included in the table) were fired with $7\frac{1}{2}$-gramme charges in blocks 15 inches square. In 9*a*, W = $2\frac{3}{4}$ inches; T = 110 lbs.; K = 49.3 cubic inches. In 9*c*, W = $3\frac{1}{16}$ inches; T = 118 lbs.; K = 29.9 cubic inches. Shot 9*b* was kept for about four months before firing, and the block thus became perfectly dried out. With W = $1\frac{3}{4}$ inches and T = 179 lbs., the whole upper surface of the block was blown off and the block was split into three pieces. It is, of course, safe to predict, in spite of this, that the value of W (for a given charge), above which no crater will be produced, will be sooner reached with a strong material than with a weak one.

Notwithstanding the insufficient range of these experiments, I think they show that the right cone cannot be taken as the shape of the crater in a homogeneous material having any considerable tensile strength; that, in such a material, the numerical relations between the crater-radius and the line of least resistance given by Hoefer's theory will not hold; and, moreover, that a complete general theory, if not practically impossible, must be much more complicated than that of Prof. Hoefer. At all events, they indicate a promising field for further investigation.

I am indebted for advice in the attempted reduction of the experiments to Prof. Mansfield Merriman, of Lehigh University, under whose direction the very laborious calculations involved were made for me by L. J. H. Grossart, C.E., of Bethlehem, Pa. The fine weighing was kindly done for me by Mr. P. W. Shimer, of Easton, Pa.

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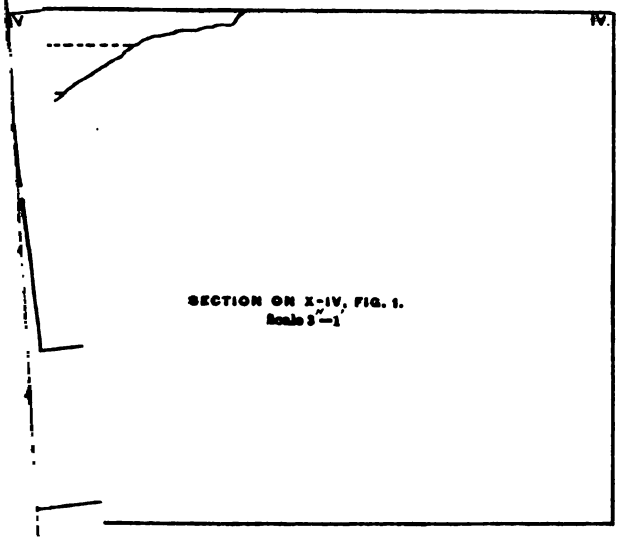
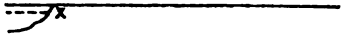
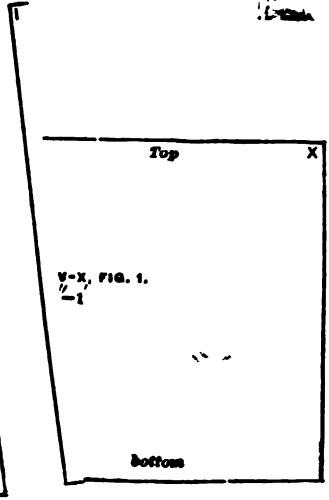


SECTION ON X-IV, FIG. 1.
Scale 3″—1′



TION ON IV-X, FIG. 1.
Scale 3″—1′



V-X, FIG. 1.
″—1′

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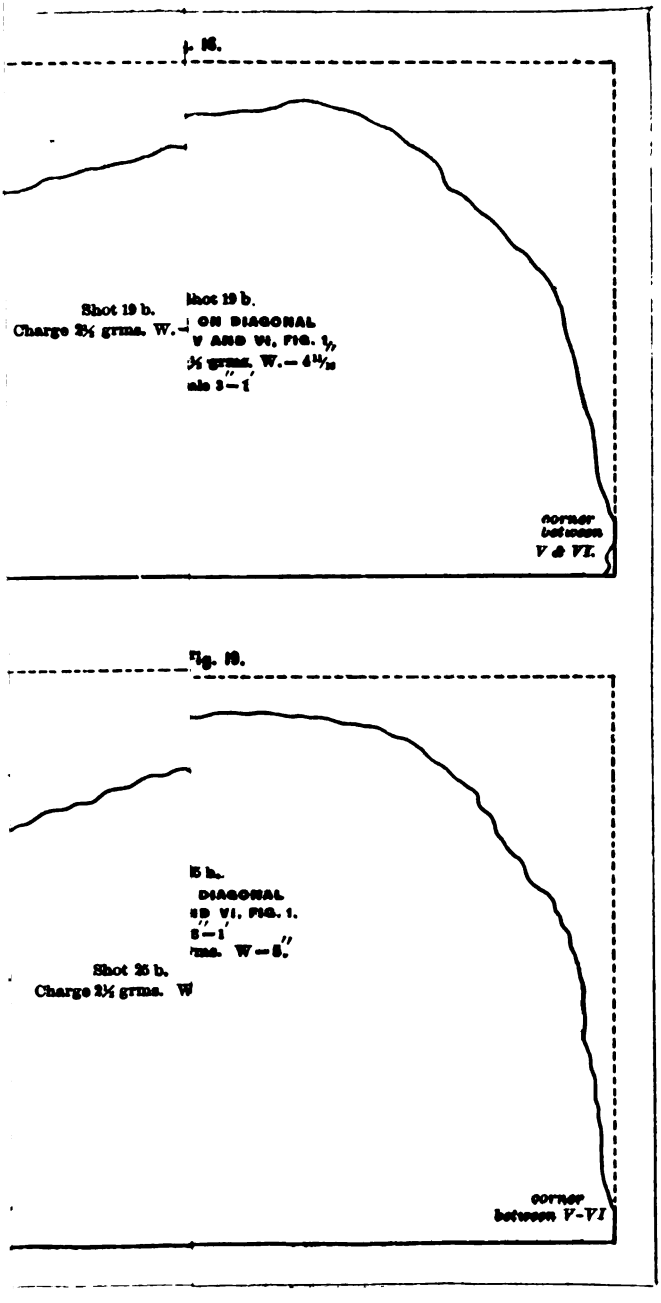
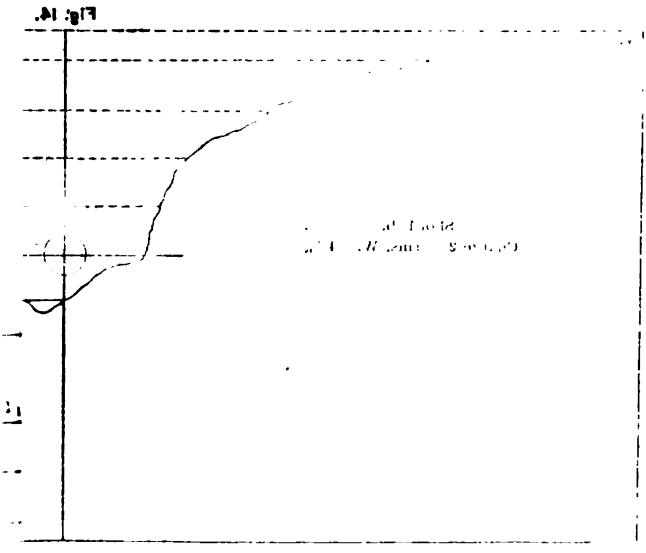
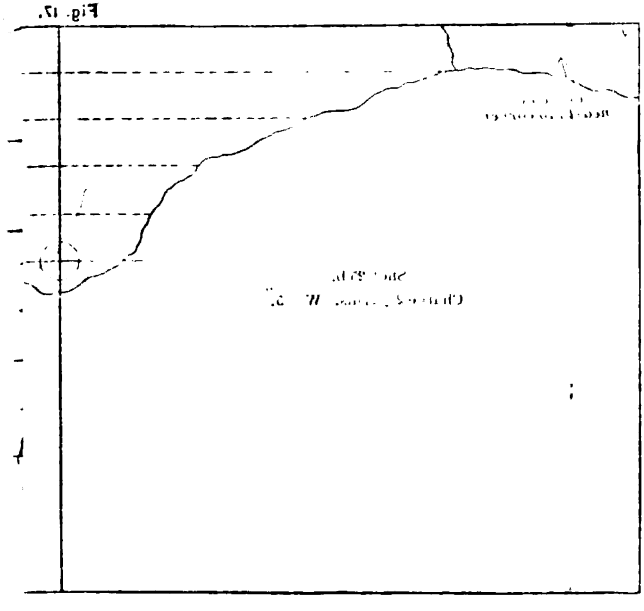

Shot 19 b.
Charge 2½ grms. W.
corner
between
V & VI.
Shot 25 b.
Charge 2½ grms. W
corner
between V-VI





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FILLING AND BLOWING-IN AT THE DURHAM BLAST-FURNACE.

BY B. F. FACKENTHAL, JR., RIEGELSVILLE, PA.

(Washington Meeting, February, 1890.)

ONE of the practical questions presented to the blast-furnace manager, with regard to which little help can be obtained from existing technical literature, is the manner of filling and blowing-in. This operation is often performed according, more or less, to traditions which have come down from earlier practice, and which, if not positively detrimental when followed with the large modern furnace, may at least be greatly improved, just as the running of the furnace itself, while in blast, has been improved. It is not necessary, as many of the old furnace-men believed, to make a lot of white or silver-gray iron at the start, or to spend many weeks, or months even, in bringing the furnace to its normal burden. It has seemed to me that the publication of carefully-recorded data of practice on this point may be interesting to the members of the Institute engaged in the management of blast-furnaces.

The enlarged blast-furnace at the Durham Iron Works, Riegelsville, Pa., owned by Messrs. Cooper & Hewitt, of New York City, was first put in blast February 21, 1876, since which time it has made seven blasts, and is now running on the eighth.

A description of the furnace-plant, together with a record of the fifth blast, was given in my paper read at the Chattanooga meeting, May, 1885 (*Transactions*, xiv., 130).

After some of the early blasts the furnace-lines underwent considerable change, and I have thought best to accompany this paper with drawings showing the lines and the tuyere-arrangement (see Plates I. and II.), and to give also the following summary of the conditions and results of each blast.

During the first and second blasts no coke was used, and the furnace ran very irregularly, showing a constant tendency to scaffold, which was attributed, in great part, to the small hearth and other defective lines. The large proportion of air required at this furnace is accounted for by the fact that the engines are defective, and do not deliver into the furnace the amount of air indicated by the piston-displacement.

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Summary of Seven Blasts of the Durham Furnace.

	1	2	3	4	5	6	7
Number of blast							
Weekly average product, gross tons	282.51	313.29	413.51	480.87	599.00	637.83	691.25
Daily " " " "	40.36	44.75	59.07	68.69	85.56	91.12	98.75
Fuel per ton of iron	1.467	1.383	1.288	1.268	1.228	1.211	1.206
Proportion of coke, per cent	None	None	11.25	19.07	18.50	23.92	34.43
Percentage of iron yielded by ores	46.82	51.38	50.57	53.79	53.33	52.53	53.65
" of limestone to ore	55.00	53.29	54.88	56.34	59.42	52.52	54.23
Cubic feet of air per minute, while running. Piston displacement	10,160	9,698	11,885	13,355	15,618	16,440	17,518
Cubic feet of air. Piston displacement for one pound of fuel	95.44	92.04	87.58	89.09	86.35	86.37	86.02
Pressure of blast	8.50	8.30	8.09	8.25	9.60	9.08	10.15
Temperature of blast, Fahr	750°	885°	868°	900°	960°	990°	958°

The principal object of this paper, however, is to give an account of the filling and blowing-in of the furnace during the six successive blasts known to us as Nos. 3, 4, 5, 6, 7 and 8. The following tables will show the filling in detail, and the results obtained during the first seven days of each blast:

Blast No. 3.

Fired up May 10, 1879, 7 P.M.

Blast put on May 12, 1879, 10 A.M. Burnt thirty-nine hours.

First cinder May 12, 1879, 5 P.M. Seven hours after blast.

First iron May 13, 1879, 1 A.M. Fifteen hours after blast.

Ore-mixture: 50 per cent. brown hematite, 25 per cent. Durham, 25 per cent. Ringwood.

Yield of ores, 46½ per cent. Fuel: ¼ coke, ¾ anthracite coal.

Charges.	Fuel, cwt.	Ores, cwt.	Limestone, cwt.	Slag, cwt.	Iron in Ores, tons.
Bed	400	.....	20	120	.....
4 each	80	50	25	20	4.65
4 "	80	55	30	20	5.12
4 "	80	60	32	18	5.58
4 "	80	65	35	16	6.05
4 "	80	70	37½	14	6.51
4 "	80	75	40	12	6.98
4 "	80	80	42½	10	7.44
4 "	80	85	45	8	7.91
4 "	80	90	47½	6	8.37
4 "	80	95	50	.....	8.84
4 "	80	100	52½	.....	9.30
Total tons,	196	165	88 8/20	30 1/2 1/2	76.75

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Pig-Iron Made, Tons.

	Silver-Gray.	No. 2.	Gray-Forge.	Mottled.	White.	Total.
May 13,	.....	.....	.....	11	.....	11
" 14,	.....	.....	.....	.....	20	20
" 15,	18½	.....	.....	6½	.....	25
" 16,	8	14	10	.....	.....	32
" 17,	.....	42	2½	.....	.....	44½
" 18,	.....	1½	29	25	.....	55½
" 19,	.....	25	35	.....	.....	60
Total,	26½	82½	76½	42½	20	248

Blast No. 4.

Fired up May 12, 1880, 6 P.M.

Blast put on May 14, 1880, 7.30 A.M. Burnt thirty-seven and a half hours.

First cinder May 14, 1880, 3 P.M. Seven and a half hours after blast.

First iron May 14, 1880, 10 P.M. Fourteen and a half hours after blast.

Ore-mixture: 50 per cent. brown hematite, 20 per cent. Durham, 30 per cent. Ringwood.

Yield of ores, 46½ per cent. Fuel: ½ coke, ½ anthracite coal.

Charges.	Fuel, cwt.	Ores, cwt.	Limestone, cwt.	Slag, cwt.	Iron in Ores, tons.
Bed	400	....	20	110	.....
4 each	80	50	25	20	4.65
4 "	80	55	30	20	5.12
4 "	80	60	32	18	5.58
4 "	80	65	35	15	6.05
4 "	80	70	37½	14	6.51
4 "	80	75	40	10	6.98
4 "	80	80	43½	8½	7.44
4 "	80	85	45	6½	7.91
4 "	80	90	47	5½	8.37
4 "	80	95	50	.....	8.84
4 "	80	100	54	.....	9.30
Total tons,	196	165	88⅛	28⅛	76.75

Pig-Iron Made, Tons.

	Silver-Gray.	No. 2x.	No. 2.	Gray-Forge.	Mottled.	White.	Total.
May 14,	.....	.....	.....	.....	10	.....	10
" 15,	.....	.....	6	7½	.....	20½	34
" 16,	26	.....	.....	.....	.....	.....	26
" 17,	20½	7½	2½	.....	.....	.....	30½
" 18,	.....	11½	38½	.....	.....	.....	50
" 19,	.....	.....	43½	12½	.....	.....	56
" 20,	.....	.....	13½	42½	.....	.....	56
Total,	46½	19	104	62½	10	20½	262½

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Blast No. 5.

Fired up March 28, 1882, 8 P.M.

Blast put on March 30, 1882, 12 M. Burnt forty hours.

First cinder March 31, 1882, 1 A.M. Thirteen hours after blast.

First iron March 31, 1882, 7 A.M. Nineteen hours after blast.

Ore-mixture: 30 per cent. brown hematite, 30 per cent. Durham, 30 per cent. Ringwood, 5 per cent. Charlotteburg, 5 per cent. scrap cinder and roll-scale.

Yield of ores, 49 per cent. Fuel: $\frac{1}{4}$ coke, $\frac{3}{4}$ anthracite coal.

Charges.	Fuel, cwt.	Ores, cwt.	Limestone, cwt.	Slag, cwt.	Iron in Ores, tons.
Bed	400	.....	20	100	.....
4 each	80	40	20	20	3.92
4 "	80	45	25	15	4.41
4 "	80	50	27½	15	4.90
4 "	80	55	30	12½	5.39
4 "	80	60	33	10	5.88
4 "	80	70	38½	5	6.86
4 "	80	75	42	.....	7.35
4 "	80	80	44	.....	7.84
4 "	80	88	48	.....	8.62
4 "	80	96	52	.....	9.41
4 "	80	100	54	.....	9.80
4 "	80	104	56	.....	10.19
Total tons,	212	172½⅝	95	20½⅛	84.57

Pig-Iron Made, Tons.

	Silver-Gray.	Gray-Forge.	Mottled.	Total.
March 31,	17½	.....	.....	17½
April 1,	.....	30	.....	30
" 2,	.....	10	27½	37½
" 3,	.....	25	27½	52½
" 4,	.....	50½	.....	50½
" 5,	.....	62	.....	62
" 6,	.....	57½	.....	57½
Total,	17½	235	55	307½

Blast No. 6.

Fired up June 30, 1885, 11.30 A.M.

Blast put on July 1, 1885, 7 P.M. Burnt thirty-one and a half hours.

First cinder July 2, 1885, 12 M. Seventeen hours after blast.

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First iron July 2, 1885, 11 P.M. Twenty-eight hours after blast.

Ore-mixture: 12½ per cent. brown hematite, 35 per cent. Durham, 30 per cent. Ringwood, 15 per cent. Elba, 7½ per cent. scrap cinder and roll-scale.

Yield of ores, 51 per cent. Fuel: ¼ coke, ¾ anthracite coal.

Charges.	Fuel, cwt.	Ores, cwt.	Limestone, cwt.	Slag, cwt.	Iron in Ores, tons.
Bed	400	.....	20	100	.....
4 each	80	30	15	20	3.06
4 "	80	35	17½	17½	3.57
4 "	80	40	20	15	4.08
4 "	80	45	25	15	4.59
4 "	80	50	27½	12½	5.10
4 "	80	60	33	10	6.12
4 "	80	70	38½	5	7.14
4 "	80	80	44	.....	8.16
4 "	80	88	50	.....	8.98
4 "	80	96	53	.....	9.79
4 "	80	104	58	.....	10.61
4 "	80	108	60	.....	11.02
Total tons,	212	161 4/20	89 4/20	24	82.22

Pig-Iron Made, Tons.

	Gray-Forge.
July 2,	11
" 3,	29
" 4,	45
" 5,	60
" 6,	65
" 7,	64
" 8,	70½
Total,	344½

BLAST NO. 7.

Fired up June 28, 1888, 8 P.M.

Blast put on June 29, 1888, 12 P.M. Burnt twenty-eight hours.

First cinder June 30, 1888, 7.30 P.M. Nineteen and a half hours after blast.

First iron July 1, 1888, 10.30 A.M. Thirty-four and a half hours after blast.

Ore-mixture: 30 per cent. brown hematite, 30 per cent. Durham, 15 per cent. Ringwood, 25 per cent. Porman (Spanish).

Yield of ores, 48 per cent. Fuel: ¼ coke, ¾ anthracite coal.

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Charges.	Fuel, cwt.	Ores, cwt.	Limestone, cwt.	Slag, cwt.	Iron in Ores, tons.
Bed	400	.....	20	80	.....
4 each	80	30	17	20	2.88
4 "	80	35	20	17½	3.36
4 "	80	40	22½	15	3.84
4 "	80	45	25	12½	4.32
4 "	80	50	27½	10	4.80
4 "	80	60	32½	7½	5.76
4 "	80	70	40	5	6.72
4 "	80	80	45	2½	7.68
4 "	80	88	50	.....	8.45
4 "	80	96	52½	.....	9.22
4 "	80	104	57½	.....	9.98
4 "	80	108	60	.....	10.37
Total tons,	212	161 4/10	90½	22	77.38

Pig-Iron Made, Tons.

	No. 2x.	No. 2.	Gray-Forge.	Total.
July 1,	.....	.....	9½	9½
" 2,	.....	.....	33	33
" 3,	15	15	12½	42½
" 4,	.....	.....	55	55
" 5,	.....	.....	60	60
" 6,	17½	37½	10	65
" 7,	47½	17½	.....	65
Total,	80	70	180	330

Blast No. 8.

Fired up March 23, 1889, 4.30 P.M.

Blast put on March 24, 1889, 9.30 P.M. Burnt twenty-nine hours.

First cinder March 25, 1889, 1.30 P.M. Sixteen hours after blast.

First iron March 26, 1889, 8.30 A.M. Thirty-five hours after blast.

Ore-mixture: 12½ per cent. Mokta (African), 45 per cent. Durham, 25 per cent. Ringwood, 17½ per cent. Porman (Spanish).

Yield of ores, 52 per cent. Fuel: ¼ coke, ¾ anthracite coal.

Charges.	Fuel, cwt.	Ores, cwt.	Limestone, cwt.	Slag, cwt.	Iron in Ores, tons.
Bed	400	.....	20	80	.....
4 each	80	30	17	20	3.12
4 "	80	35	20	17½	3.64
4 "	80	40	22½	15	4.16
4 "	80	45	25	12½	4.68
4 "	80	50	27½	10	5.20

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Charges.	Fuel, cwt.	Ores, cwt.	Limestone, cwt.	Slag, cwt.	Iron in Ores, tons.
4 each	80	60	32½	7½	6.24
4 "	80	70	40	5	7.28
4 "	80	80	45	2½	8.32
4 "	80	88	50	.....	9.15
4 "	80	96	52½	.....	9.98
4 "	80	104	57½	.....	10.82
4 "	80	108	60	.....	11.23
Total tons,	212	161 4/10	90 1/2 8/10	22	83.82

Pig-Iron Made, Tons.

	No. 2.	Gray-Forge.	Total.
First day,	.....	7½	7½
Second day,	7½	17½	25
Third day,	.....	37½	37½
Fourth day,	.....	62½	62½
Fifth day,	.....	70	70
Sixth day,	.....	67½	67½
Seventh day,	.....	70	70
Total,	7½	332½	340

For convenience in keeping the office-records, our charging-scales all have gross beams, and the weights of fuel, ore and limestone are designated in our books (as well as in the above tables) as cwt., instead of being given in pounds, as I believe is usually done elsewhere.

After putting in the wood and "bed" of fuel, which I will describe later on, we commence using regular charges, and at once establish our unit of fuel, which, during the six blasts, has remained at 80 cwt., or 4 gross tons. My practice in starting a furnace has been, as the tables show, to increase the burden at the end of every four charges.

The filling for the third and fourth blasts was practically the same, differing only slightly as to the proportion of blast-furnace slag used. The ore-mixtures were not identical, but the yield in both cases was the same, viz., 46½ per cent. Both started (after putting in the bed) with a burden of 50 cwt. of ore, and were increased 5 cwt. at the end of every fourth charge until the furnace was full with a burden of 80 cwt. of fuel to 100 cwt. of ore, or one unit of fuel to one and one-quarter units of ore. The results obtained from these two fillings were almost the same. No. 3 produced 248 tons and No. 4 produced 262½ tons during the first seven days. In both cases the first and second casts were mottled and white iron, while the third and fourth casts were silver-gray iron, after which the

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product was No. 2 and gray-forge. The mottled iron made on the sixth day of the third blast was the result of a stop. As I prefer to blow in on open gray-forge iron, I endeavored, as far as possible in filling for the fifth blast, to correct the mistakes of the third and fourth fillings, and to avoid making mottled and white, as well as silver-gray, iron. To correct the former tendency I started with a lighter burden, and by increasing the burden faster at certain stages of blowing-in I sought to avoid making silver-gray iron. I started with 40 cwt. of ore, and ended with the furnace full at 80 cwt. of fuel to 104 cwt. of ore. The results show that while I thus avoided making white iron, I continued to make some silver-gray (17½ tons), and a considerable quantity (55 tons) of mottled. The working of the furnace clearly indicated that the mottled iron was caused by an overburden. Moreover, the ore-mixture for the fifth filling contained less hematite ore, which may have been an additional cause of the production of this mottled iron. To correct this filling still further, I decided, in filling for the sixth blast, to commence with a still lighter burden than for the fifth, and to increase faster, ending with the furnace full at 80 cwt. of fuel to 108 cwt. of ore. This gave me the desired results. The first cast was good, strong gray-forge, a little high in silicon, and the product for some days continued to be gray-forge; and although we made a considerable proportion of foundry-iron during the early weeks of the blast, only 30 tons of mottled was made up to the eleventh week.

As this filling and blow-in had given us satisfactory results, I made no changes in filling for the seventh and eighth blasts, desiring, if possible, to confirm by repetition the policy of the practice pursued. The tables will show the degree of uniformity in the result. In the seventh blast our first cast, as before, was gray-forge, rather high in silicon, but, by reason probably of a more favorable ore-mixture, we produced some No. 2 and No. 2x during the first week. In the eighth blast, the first and second casts were gray-forge, while the third cast contained 7½ tons of No. 2 and 5 tons of gray-forge, and for the remainder of the week the grade was gray-forge.

In filling a furnace I always put a scaffold or cribbing of wood up to the tuyeres, leaving an open space underneath. I think this is better than placing the wood directly on the hearth. Kindling wood is then put in front of the tuyere-openings, and the furnace is lighted all around. I prefer not to light the furnace at the front. Our furnace has a closed front, which is walled in before lighting. My objection to putting wood in the bottom of the furnace below the tuyeres

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is, that in my experience I find no air can reach it, and it is likely to become covered with ashes and remain there for some time without burning. If a scaffold is put in with care, there is no special danger of any irregular settling of the stock. At the Durham furnace, with a hearth-diameter of 11 feet 4 inches, a space, say 4 feet 6 inches high, underneath the scaffold, is equivalent to only 18 inches of height by the area in the large part of the furnace. Hence, there can be no unusual movement of the stock when the scaffold burns away. Moreover, I think it is an advantage to have the stock move somewhat while the wood is burning up. It doubtless lessens the danger of scaffolding during the blowing-in, and I am not sure but that the wood scaffold in the hearth is an advantage for this reason. Immediately on the top of the scaffold I put a layer of cord-wood, placed horizontally; on the top of this, three tiers of wood resting on the ends; and on the top of this, the bed of fuel. Fig. 8, Plate II. shows a section of the furnace as it was filled for the sixth, seventh and eighth blasts. In this figure, the packing between the bricks next the shell and the in-wall, is not as clearly shown as in Fig. 7; otherwise, the drawing correctly represents the furnace as well as the filling.

A study of the successive forms given to the interior of the Durham furnace, and the arrangements of hearth and tuyeres in connection with the results of the successive blasts, is very instructive.

The following notes, made during the first week of the eighth blast, will, I trust, prove interesting as showing the behavior of the furnace until the regular burden came down:

Furnace-Notes.

In filling a furnace it often happens that part of the stock is more or less wet. Even when dry stock is filled, and particularly when it is left standing, if only for a day or two, while the charging apparatus is being adjusted, or for any other reason, more or less moisture is absorbed, and it would require some hours to kindle the furnace and get a natural draft through it.

To save time, therefore, and to guard against an irregular burning, I put up the tuyere-pipes (through the openings in the water-breasts or tuyere-coolers) and by means of the blast-engines blow a gentle pressure of air, which soon clears away the smoke and creates a natural draft. Before lighting the furnace I close the iron-notch and cinder notch. The furnace is then fired at all the tuyeres.

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For our No. 8 blast the furnace was lighted March 23, 1889, 4.30 P.M.

5 P.M.—Put up tuyere pipes and started engine.

6 P.M.—Smoke appeared at tunnel-head.

7 P.M.—Strong draft at tunnel-head; tuyeres bright; stopped blowing.

From this time, until the furnace was ready for blast, the tuyeres were watched with care, kept free from ashes, and regulated so that the draft through them would be uniform all around. Wind blowing from the west necessitated putting a brick or two in the tuyeres on that side. During the burning of the furnace the charging-bells were kept open.

March 24, 7 P.M.—Cinder dripping quite freely at all tuyeres. An inspection at the iron-notch and cinder-notch showed a considerable amount of melted cinder, and preparations were made to put on the blast. Cinder-notch and tuyeres were put in, and the iron-notch was cleaned out and stopped with coal-dust.

9.30 P. M.—Blast put on, the furnace having burned twenty-nine hours. Used seven three-inch nozzles. Started by blowing 6000 cubic feet of air per minute, through two iron-pipe stoves of the Durham pattern. Pressure very low, but not exactly measured, the gauge not registering pressures less than half a pound. Valves on cold side of ovens, almost closed, in order to get engine-room pressure enough to close and operate the charging-bells. Gas was abundant from the start; but tuyeres were not bright and required considerable pricking. Stock had settled 7½ feet in burning up.

11 P.M.—6500 cubic feet of air. Pressure, ½ pound. Temperature, 480° F. Tuyeres brighter. Commenced filling.

12 P.M.—7000 cubic feet of air. Pressure, ⅝ pound. Temperature, 600° F.

March 25, 3 A.M.—7500 cubic feet of air. Pressure, ¾ pound. Temperature, 680° F.

6 A.M.—8000 cubic feet of air. Pressure, 1¼ pounds. Temperature, 760° F. Tuyeres very bright; stock settling regularly. Filled 3 charges during night, each 80 cwt. fuel, 108 cwt. ore, 60 cwt. limestone. Stock down 7 feet.

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9 A.M.—8000 cubic feet of air. Pressure, 1¾ pounds. Temperature, 860° F. Gas continued to get richer. Fires under boilers put out.

12 M.—8000 cubic feet of air. Pressure, 2¼ pounds. Temperature, 900° F. Could easily carry more, but, as heat was not needed, I thought best to keep a reserve. Turned water into bosh-coils Steam blowing off at boilers.

1.30 P.M.—First cinder sixteen hours after putting on blast. About 3 tons; color gray; a little stringy; tapped through bronze-notch with 1-inch opening. Cinder contains 39.73 per cent. silica.

3 P.M.—8500 cubic feet of air. Pressure, 3¼ pounds. A third hot-blast oven put on in order to dispose of surplus gas.

4.30 P.M.—Second cinder, 3 tons. About the same as first tap.

6 P.M.—8500 cubic feet of air. Pressure, 3¼ pounds. Temperature, 860° F. Cinder getting hotter.

12 P.M.—9000 cubic feet of air. Pressure, 3½ pounds. Temperature, 860° F.

March 26, 3 A.M.—9000 cubic feet of air. Pressure, 3¾ pounds. Temperature, 880° F.

6 A.M.—9500 cubic feet of air. Pressure, 4 pounds. Temperature, 880° F. Furnace full. Stock filled during past twenty-four hours, 5 charges, each 80 cwt. fuel, 108 cwt. ore, 60 cwt. limestone; 6 charges, 80 cwt. fuel, 112 cwt. ore, 62 cwt. limestone.

8.30 A.M.—12 cars of cinder out. Silica in twelfth car, 39.98 per cent. Made first cast, 7½ tons gray-forge. Stock settling regularly and evenly. No slips at cast. Changed nozzles from 3-inch to 3½-inch.

12 M.—9500 cubic feet of air. Pressure, 5 pounds. Furnace getting grayer; not driving fast enough. Commenced filling some scrap through inside charging-bell.

3 P.M.—10,000 cubic feet of air. Pressure, 5½ pounds.

4 P.M.—Furnace slipped 3 feet, evenly all around, while blast was on. Pressure before slip, 5½ pounds; after slip, 4½ pounds. Burden at work, 45 cwt. of ore.

6 P.M.—11,000 cubic feet of air. Pressure, 4½ pounds.

9 P.M.—11,000 cubic feet of air. Pressure, 5 pounds.

11 P.M.—Second cast, 12½ tons gray-forge. Furnace slipped 3 feet, when blast was taken off. On starting again, pressure was 4 pounds. Burden at work, 60 cwt. of ore.

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March 27, 3 A.M.—11,000 cubic feet of air. Pressure, 4½ pounds.
6 A.M.—Third cast, 7½ tons No. 2 and 5 tons gray-forge. Furnace regular. Pressure, 4½ pounds, both before and after cast. Burden at work, 70 cwt. of ore. Stock filled during past twenty-four hours, 14 charges, each 80 cwt. fuel, 112 cwt. ore, 62 cwt. limestone, 12 barrows (about 5 tons) scrap.
9 A.M.—11,500 cubic feet of air. Pressure, 4½ pounds.
1 P.M.—Fourth cast, gray-forge. Cinder getting darker.
3 P.M.—12,000 cubic feet of air. Pressure, 4¾ pounds. Temperature, 980° F.
7 P.M.—Fifth cast, gray-forge. Cinder improving.
9 P.M.—Pressure and other conditions the same.
12 P.M.—Pressure, 5 pounds. Cinder gray.
March 28, 6 A.M.—Stock filled during past twenty-four hours, 18 charges, 80 cwt. fuel, 116 cwt. ore, 65 cwt. limestone. 12,000 cubic feet of air. Pressure, 4¾ pounds.
March 29, 6 A.M.—Stock filled, 19 charges. Same burden. 13,000 cubic feet of air. Pressure, 5½ pounds.
March 30, 6 A.M.—Stock filled, 21 charges. Same burden. 13,500 cubic feet of air. Pressure, 6 pounds.
March 31, 6 A.M.—Stock filled, 20 charges. Same burden. 13,500 cubic feet of air. Pressure, 6½ pounds.
April 1, 6 A.M.—Stock filled, 22 charges. Same burden. 14,000 cubic feet of air. Pressure, 7½ pounds.
April 2, 6 A.M.—Stock filled, 23 charges. Same burden. 14,500 cubic feet of air. Pressure, 8½ pounds. Furnace getting grayer. Iron, No. 2.
April 3, 6 A.M.—Stock filled, 23 charges. Burden increased to 120 cwt. ore, 68 cwt. limestone. 15,000 cubic feet of air. Pressure, 8½ pounds. Iron, No. 2 and No. 2x.
April 4, 6 A.M.—Stock filled, 22 charges. Same burden. Nozzles enlarged from 3½ to 4 inches. Iron, 77½ tons, mostly No. 2x.
April 5, 6 A.M.—Stock filled, 24 charges. Same burden. 15,500 cubic feet of air. Pressure, 8½ pounds. Iron, 80 tons No. 2x, No. 2 and gray-forge.
April 6, 6 A.M.—Stock filled, 26 charges. Burden increased to 124 cwt. ore, 70 cwt. limestone. 16,000 cubic feet of air. Iron, 90 tons No. 2 and gray-forge.

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PLATE II.—LAMTEXTHAL.

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April 7, 6 A.M.—Stock filled, 24 charges. Same burden. 16,500 cubic feet of air. Nozzles enlarged from 4 to 5 inches, being the full area of the tuyere, at which they remained throughout the remainder of the blast.

The only irregularities of any kind observed during the blowing-in were the two slips, each of 3 feet, at 4 P.M. and at 11 P.M. on March 26th.

The largest volume of air used at any time during the blast was 20,000 cubic feet per minute, but some allowance should be made from this and from all the air-measurements which I have given for defective engines, on account of which the loss in delivering the air was excessive.

The output during the first regular week of this blast was 555 tons; the second week, 630 tons; the third week, 750 tons; and the fourth week, 770 tons. The largest output for any one week was reached during the thirtieth week of the blast, when 915 tons were made of No. 2x, No. 2 and gray-forge.

NOTES ON THE GOLD-DEPOSITS OF MONTGOMERY COUNTY, MARYLAND.

BY S. F. EMMONS, U. S. GEOLOGICAL SURVEY, WASHINGTON, D. C.

(Washington Meeting, February, 1890.)

Introductory.—In view of the fact that an opportunity is to be offered to the members of the Institute present to inspect the recent developments of gold-deposits in the immediate vicinity of Washington it has seemed appropriate that some preliminary description of them should be given which may facilitate that inspection.

It would have been a matter of much satisfaction to me to be able to present some general description of the gold-deposits of the Appalachian system; but I have as yet had no opportunity of personally examining any others than those of Montgomery county, and these are still in their infancy. It is, moreover, a slower and more difficult operation to make geological observations in this region than in the more northern States; for whereas in the latter region the

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ice-sheets of the Glacial period have scored off the surface of the whole country, in geologically recent times, down to a base of the hardest and most unaltered rock-masses, here the rotting of the rocks under the influence of atmospheric agents has been going on uninterruptedly for a long period, which cannot be measured by years but which may reach back into Mesozoic time. Hence in areas formed of rocks which, like those which carry the gold, stand almost vertically, and which by their mineralogical composition are peculiarly susceptible to decomposition by surface-waters, it is only in the beds of the larger streams that rocks are found in sufficiently fresh condition to permit an approximate determination of their original character. On the intermediate ridges, plateaux and hills one may often sink for a hundred feet through picking-ground before reaching hard rock, and there is a gradual passage from the red clayey soil that forms the upper ten or twelve feet into material which, on a fresh face, still shows the schistose structure of the original rock, but which crumbles on exposure to the air.

It seems to be a matter of surprise to many that gold in paying quantities should be found so close to the national capital; but there is nothing in the geological history of the region which renders it improbable.

From Harper's Ferry to Washington the Potomac river runs across a belt of crystalline rocks which widen both northward and southward of the river. In Virginia the eastern boundary of this belt runs almost due south from Washington to Richmond and beyond, while its western boundary follows the prevailing northeast and southwest trend of the Appalachian system along the eastern side of the valley of Virginia. Within this belt, but more especially in the middle eastern portion, in Spottsylvania and adjoining counties, gold was known to exist as far back as Colonial times. Prof. William B. Rogers,* whose early investigations into the geology of Virginia still form our most reliable source of information with regard to a great portion of the state, devotes considerable space in his report of 1836 to a description of the gold-bearing belt, which is quite applicable, as far as it goes, to the deposits of Montgomery county, Md. In this, while deprecating any over-wrought anticipations of their productiveness, he advances the opinion that their working is destined under proper management "to become an important branch of the systematic industry of the State."

It is only since 1829 that any record of actual production from

* *Geology of the Virginias*, p. 132.

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mining operations in Virginia seems to exist. The U. S. Mint returns from 1829 to 1854 show that nearly a million and a half of gold was deposited during that period from Virginia mines. During the war all mining operations were necessarily interrupted, but, under the inflated and speculative conditions that followed it, mining operations were resumed and considerable Northern capital was invested in reduction plants. The greater number of these have for some reason or other been closed down and practically abandoned until within the last year or two. From the little I have had opportunity of learning about them, I think it may fairly be assumed that on the whole the failures have been in a large measure due to bad management, to inexperience in mining, and to a want of proper understanding of the geological relations of the deposits. The product of Virginia in these later years has not always been segregated in the published mint-returns, so that its actual amount is not readily determinable, but it has probably not exceeded ten or fifteen thousand dollars a year.

On the north side of the Potomac in Maryland, gold does not appear to have been much sought for. The only record I find of its discovery in early days is in the *Proceedings* of the American Philosophical Society for April, 1849, where mention is made of the occurrence of veins of quartz carrying gold on the farm of Mr. Samuel Ellicott, in Montgomery county, thirty miles west of Baltimore, three specimens of which assayed respectively $610, $787, and $168 per ton. This is accompanied with a short geological sketch of the locality by Prof. E. Emmons, of Albany, which would be equally applicable to the veins of which I shall speak to-day, though he does not seem to have recognized the fact that the soft clayey rock in which the gold was found was only the surface-alteration of the underlying crystalline rocks, for he apparently regards them as surface-deposits and recommends their working by open cuts or trenches. I am unable to say exactly where Mr. Ellicott's farm was situated or whether the deposit was ever practically exploited; but I am inclined to think that the recommendations of my namesake would, under existing conditions, naturally have led to disastrous financial results.

Geological Relations.—Before proceeding to a local description of the region about to be visited it may be well to give a brief outline of its broader geological relations.

The geological section across the Appalachian system in this latitude is one of the most typical and instructive to be found along

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its entire length. West Washington or Georgetown, as you know, is situated at the head of tide-water and of navigation on the Potomac river. Above this point the river runs rapidly in a relatively narrow rocky valley or cañon, from 300 to 400 feet below the adjoining country, which is a gently-rolling table-land cut by deep ravines, locally known as "runs." Below Georgetown it suddenly widens out into a broad stream, often more than a mile wide, flowing sluggishly through a generally low and more level country, though bluffs along the river often rise 100 feet above its level.

The site of Washington marks an even more pronounced change in the geology than in the topography of the regions respectively above and below it. It is on the shore-line of an ancient ocean whose waves, from late Jurassic far down into Tertiary times, beat against the slopes of the table-land above, then much higher than it is to-day. In this ocean there was deposited from the abrasion of the land a series of gravels, clays and sands, which form the lowland coast belt from New Jersey southward to Florida.

In almost any of the fresh cuts made by roads leading out from the northwestern part of the city to the flat-topped ridge running northward from Georgetown to Tenallytown, or in the ravines leading down to the deep valley of Rock creek, which runs southward along the eastern flanks of this ridge, we may distinguish a bed of conglomerate, resting unconformably on an uneven and eroded surface of more or less decomposed crystalline schists. This is the basal conglomerate of the Potomac formation, of late Jurassic or early Cretaceous age, which runs from here northeastward to Baltimore, with beds of fine clays and sands, carrying in places workable deposits of iron-ore, resting upon it, which in turn further eastward are covered by the Tertiary beds that form the general surface of the Atlantic coastal region. The evenly-rounded character of the pebbles of this conglomerate, which are composed of the hardest and toughest kind of quartzite, show that they have been subjected to a long process of trituration in water; and this, together with their position, indicates that they probably constituted the beach-line of an ancient continent. The outlines of this shore have not yet been traced in detail: it probably reached a little to the westward of the high ridge I have spoken of, which runs north from Georgetown. The uses of these pebbles or boulders are various. In modern times they have furnished most of the paving-stones used in the city of Washington; and more anciently, as has been shown by recent investigations by members of the Bureau of Ethnology, they were

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extensively quarried just north of the city by the tribes of Indians, who once occupied this region, for the purpose of making out of them the arrow- and spear-heads, hammers and axes, and other stone implements which were in use among them before the advent of the white man. Their petrological character and certain fossil markings found in them show, moreover, that they must have been derived from the beds of Cambrian quartzite, which now are first found nearly fifty miles up the river, along the borders of the valley of the Shenandoah.

At various points in the table-land region, and still more frequently in the broader portion of the crystalline belt to the southward, are found more or less extensive patches of red sandstone of Triassic age, resting on the upturned edges of the crystalline beds, generally in a nearly horizontal position. One of the best-known localities is the Seneca quarries on the north bank of the Potomac river, some twenty miles above Washington, which furnish the deep-red freestone, much used as a building-stone in this city. These sandstones take us back a step further in the geological history of the region. They indicate that, at some time previous to the early Jurassic, the greater part of the crystalline belt was submerged beneath the ocean. It is possible, indeed, that the waters extended over the whole of it; but the detached position of these bodies of sandstone and their general appearance of filling synclinal troughs or depressions affords some ground for the supposition that higher ridges projected above the water between them, which have since been planed down by erosion.

The belt of crystalline rocks along the Potomac river is about fifty miles in width. Beyond it, commencing in the neighborhood of Harper's Ferry, is a series of distinctly-bedded sedimentary rocks, consisting of quartzites, limestones, slates, sandstones and conglomerates, and extending in geological succession from the Cambrian up to the top of the Carboniferous. This series, closely compressed into a complicated system of folds and faults, forms the main elevation of the Appalachian mountain system. Its aggregate thickness is estimated to be in its greatest development from 6½ to 7 miles, but this thickness decreases rapidly to the westward toward the Mississippi, being less than 4000 feet in Missouri. As these beds are largely fragmental, it is evident that they must have been formed from the abrasion of some continental land-mass; and the composition and present distribution and position of the beds show that this land-mass lay to the eastward; the belt of crystalline schists must

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have constituted part of this land-mass or continent, a great part of which, however, as is shown by other geological deductions—which it were too long to enter into here—must have sunk beneath the level of the ocean during the many geological movements that have supervened since Carboniferous time.

With regard to the crystalline rocks, with which we are now principally interested, and the formation of which dates back into still more remote periods of geological history, we have yet but little definite knowledge beyond the fact that they are probably pre-Cambrian. By Rogers and others of the earlier geologists they were simply classed as Azoic or Archæan, though he recognized certain elementary lithological distinctions in their petrological characters. Hunt and others of his school have attempted to divide them into Laurentian, Huronian and Montalban, on grounds of general lithological correspondence to rocks which are typically developed in far distant portions of the country,* but general deductions on such a basis, in rocks which have evidently suffered such complete metamorphism as these have, are at best very uncertain. It will only be after the field has been accurately and systematically examined, and the metamorphic changes in the rocks have been traced with the aid of microscopical study by trained petrologists, that even the relative age of the different rocks can be definitely determined.

Rogers recognized in Virginia, along the eastern border of the belt, gneisses and granites which, from his description, seem to correspond in some degree to the Laurentian rocks of the north. The auriferous veins, according to him, occur in a series of micaceous, talcose, chloritic and garnetiferous slates, with which are associated in certain localities siliceous beds, roofing-slates and limestones.

J. D. Whitney says of the gold-belt from Georgia to Pennsylvania:† "Throughout this whole extent, the auriferous belt presents rocks of nearly the same character; they are slates of every variety, intermixed with bands of a granitic and syenitic character. The predominating kind of slate is talcose, passing into chloritic and argillaceous. The prevailing dip is east at a very high angle. In Virginia they stand nearly vertical."

My own personal obeservations have not extended south of the line of the Potomac, so that I can only judge from the somewhat

* *Trans.* x., 477. C. H. Hitchcock, from analogies with New England rocks, regards the Potomac belt as of Huronian age, but seems doubtful whether it is above or below the Montalban.

† *The Metallic Wealth of the United States*, p. 122. Lippincott. 1854.

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meager published data of others; but the evidence, so far as it goes, seems to indicate that the gold-bearing rocks along the whole length of the Appalachians belong to one general system and are much later in age than the true Archæan. Gold has thus far been most abundantly produced from the southern extremity of the belt. In the Middle States but little has been found, and it is by no means certain that exactly the same series of rocks is represented. In Nova Scotia, on the other hand, there has been a small but steady production of gold since 1862.* The rocks in which it occurs there, are pronounced by Dawson, on somewhat meager palæontological data, to be of lower Cambrian age. They are less highly metamorphosed and in general more siliceous than those of the Appalachian belt; but the manner of occurrence of the quartz-bodies and of the gold presents many analogies with the latter, and it is within the bounds of possibility that both series may ultimately be found to be of nearly corresponding age.

The rocks of the auriferous belt along the Potomac river, from Washington to the bend of the river at Great Falls, a distance of about 16 miles, are remarkably uniform in character. In the most unaltered form, they appear to be fine-grained micaceous schists, rather massive than schistose in structure; that is, while there is a distinctly laminated or banded texture in the rocks, due to the parallel arrangement of the constituent minerals, especially the mica, the bedding-planes, if they ever possessed any, have become obliterated by metamorphism, or their individuality is lost to the eye amid the multitude of cross-fractures or cleavage-planes. The lamination is, however, very uniform in direction, having a general north and south strike and a dip that is nearly vertical or very slightly inclined to the eastward. Whether it is parallel to the original bedding, or is simply the result of great lateral pressure, still remains to be determined.

The rock exposed in the bed of the Potomac, is naturally the freshest, since it lies deeper below the general surface of the plateau than that exposed by any ravine or shaft-cutting. It is quarried along the Virginia side of the river, from the Aqueduct to the

* Gilpin, *Trans.*, xiv, 688. Production of gold in Nova Scotia, from 1862 to 1885, 388,596 ounces. It is interesting to note, illustrative of the greater efficiency of labor, after experience in minin- duction has been gained, that according to Mr. Gilpin's table, the product of 1862, viz., 7275 ounces, was the result of 156,000 days' labor, while that of 1885, viz., 21,600 ounces, was the result of only 156,361 days' labor.

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Chain bridge, for foundation-stone. This rock consists mainly of quartz and muscovite, with a varying quantity of biotite: besides these, it contains orthoclase and garnet, with a chloritic mineral and microscopic ore-particles. A somewhat coarser variety, approaching a gneiss in structure, is also quarried on the Broad branch of Rock creek and makes a handsome gray building-stone. In most of the rock-cuts away from the river, however, one finds only a more or less decomposed rock-mass, in which, on a fresh surface, the original lamination of the rock can be distinguished; but on exposure to the weather the surface disintegrates and is soon no longer distinguishable from any ordinary clay-bank. In traversing this portion of the country, one remarks the extraordinary abundance of angular fragments of white vein-quartz. The macadamization of all the principal roads is done with this material; and, in the region west of the old shore-line above mentioned, all the stones in the fields not yet cleared by the farmers are fragments of vein-quartz and are often nearly as abundant as in the glaciated regions to the north, though, as distinguished from those, never rounded but always sharply angular. It is a matter of speculation how, in a region where the surface-rock has so evidently rotted in place, without being disturbed or rearranged, these rocks came to be so widely distributed as they often are. Professor W. C. Kerr* has proposed an extremely plausible explanation for similar phenomena in North Carolina, that the distribution is due to the action of frost, from moisture penetrating the soil during the winter months. It is hence called by him "frost-drift."

Gold-Developments.—To what extent the numerous quartz-veins in the immediate vicinity of Washington are gold-bearing is yet undetermined, what little prospecting is done being generally carried on by panning the stream-beds and soil. It is known, however, that some contain appreciable amounts of gold, and no doubt traces might be found in almost all of them. The principal developments of gold, however, have been made in the neighborhood of Great Falls, about 15 to 16 miles west of the city. Report says that when the California volunteers were camped in this neighborhood during the war, some of them discovered gold on one of the farms, and promised to return when the war was over and develop it; but it is supposed that those who made the promise did not survive the war, as it was never carried out. Some twelve or fifteen years since, a mine was started in this region on the upper part of Rock Run, a

* *Trans.*, viii, 462.

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small stream that empties into the Potomac from the north, just below the Great Falls. It was known as the Montgomery mine. On its grounds now stand, uncovered and in a deplorable state of ruin, a Fraser & Chalmers 10-stamp mill, hoisting-engine, boilers to each, two pans and settlers, a reverberatory furnace, and a shaft with a considerable dump-pile, part of which has fallen into the shaft, all bearing silent testimony to an untimely demise of the undertaking. I have no definite information as to the cause of its death, but there is some reason to assume that "patent processes," a disease common to newly-opened mining districts, had something to do with it. Recently, a new organization, called the Potomac Mining Company, has been formed, and some new shafts have been sunk on the vein, but no work is now being done, pending a series of experiments to determine what form of crushing-machinery it is advisable to put up, the owners having a prejudice against stamps.

Mr. A. B. Russ, the president of the present company, informs me that he holds mint-receipts for nearly $8000 worth of gold taken out by the old company, mainly from the bed of the stream which runs through the property; and I have seen many fine nuggets, weighing up to 3 or 4 ounces, which are said to have been obtained from it quite recently.

Another abandoned mine, which, however, I have not seen, was called the Maryland mine, and is said to have been opened as early as 1867. This is situated a short distance north of the Conduit road, about half-way between the Great Falls and the Harrison property.

In the spring of 1888, Mr. Kirk, a Georgia miner, discovered gold on the farm of Mr. Harrison, about a mile south of the Montgomery mine, and, after prospecting the ground and the adjoining streams with the gold-pan, secured an option on the farm, and proceeded to prove the veins by a series of open pits. This property, in the following summer, passed into the hands of Dr. W. Kempster and associates, by whom it was slowly but systematically developed. A little 3-stamp mill was put up, in which the quartz from the various veins that ran through their grounds was practically tested, and found to run from $12 to $30 per ton, the veins being taken from wall to wall. A great deal of quartz, showing free gold, was also obtained from the veins, but this was discarded from the working-tests, which were directed to obtaining the average yield of ore, showing no free gold. Sluicing was also attempted in the bottom land of Rock Run; but, although after stripping the soil good pay

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gravel was found on the bed-rock, the slope of the valley was found to be insufficient for advantageous work, and it was abandoned for the time being.

In July, 1889, a 10-stamp mill from one of the abandoned properties in Virginia was put up on the Harrison property, and a vertical shaft sunk on one of the veins; underground development has since been carried on in a systematic manner with a small force of men, the plan of the manager of the property being to make the developments pay for themselves. Under the stimulus of the developments on the Harrison property, the adjoining country has been extensively prospected during the past year and a half, and already a number of companies have been formed, and three small reduction-plants, besides that on the Harrison mine, have been erected. Underground developments on these properties are as yet, however, extremely limited, none of their shafts, so far as I know, being more than 50 feet deep.

The general distribution of those properties which I have visited is as follows, commencing at the west:

The Allerton-Ream Property.—This is situated on the east bank of the Potomac, where the river runs nearly north and south, about three-quarters of a mile above the Great Falls. The deposit has been traced in a north and south direction across two spurs and an included ravine, extending westward from the higher interior region toward the bottom land of the Potomac river. In the bottom of the ravine near its mouth, which is but little above the level of the river, the country-rock, a curly or somewhat contorted mica schist, is exposed in a comparatively unaltered condition. Upon this rock-foundation, at the mouth of the ravine, is situated the mill, and the ore-deposit crosses the ravine about 100 yards above it. It is at present being worked by open cuts from the ravine-bottom, which cut the spurs about 50 feet below their summits. At this depth the rock is still so decomposed that it is difficult to judge of its structure, but the gold-bearing portion appears to be a belt from 4 to 10 feet wide, adjoining a fault-plane, which has the regular north and south strike of all of the veins in the region. No vein-quartz appears in the open cut, but on the surface of the spurs croppings show the peculiar honeycombed quartz, known to the miners as "hickory bark," which is characteristic, according to Rogers, of the auriferous belt in Virginia. It is a more siliceous band of schist carrying pyrites, which have been oxidized and more or less completely dissolved out. Such rock is always relatively richer than other portions of the de-

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posit. Mr. E. N. Riotte, who is consulting engineer, estimates that this deposit will run about $5 per ton, at which tenor, owing to its soft nature and to the easy transportation over a short tramway to the mill, it ought to be worked at a profit. Mr. Riotte estimates the cost of mining and milling at less than $1 per ton. Only a few hundred-ton lots have yet been crushed, and these were not considered fair working-tests, as the machinery was not in good working-condition. The reduction-plant is a Wiswell mill, a sort of cross between an arrastra and a Chilian mill, consisting of a large circular pan 10 or 12 feet in diameter, in which run four iron wheels, attached to arms of a vertical shaft. These wheels have a V-shaped circumference, and run in correspondingly-shaped grooves in the bottom of the pan. They weigh 3000 pounds each, and crush by their weight alone. A Gates crusher was being introduced for preliminary crushing, in place of the Blake crusher which had formerly been used.

On the Ford farm, of which this property formerly formed part, on the higher ground near the farm-house and perhaps a quarter of a mile west of these workings, five north and south quartz-veins, carrying gold, have been proved by shallow pits, which it is said to be the intention of Mr. Ford to open and develop.

Harrison Group.—About 2½ miles east of this, and a mile north of the Conduit road, is the group of deposits around the Harrison mine. The Conduit road, be it said in parenthesis, is one built by the government along the north bank of the Potomac on the Aqueduct, which was constructed just before the war to bring the water of the river from Great Falls to the city of Washington. It is largely macadamized with gold quartz from the adjoining farms, some of which is said to assay as high as $16 per ton.

The land in this region forms part of an original grant from the Crown, made in Colonial times, and of some of the farms only one or two transfers are recorded, since this original grant, up to present date. Mr. Harrison, former owner of the farm on which Kirk first found gold, belonged to the Virginia branch of the family of that name, from which the present President of the United States is descended.

The Harrison property consists of several farm-tracts, aggregating together about 340 acres, in a rather irregular form, through the northern portion of which runs the valley of Rock Run, which, in this portion, has a nearly east and west direction. Adjoining and interlocking, as it were, with this, are the properties of the Eagle and Irma mines, on the west and east sides respectively, each of which

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has put up a 10-stamp mill within the past six months. As the veins worked on each of these properties run through part of the Harrison property, a description of those in the latter will include them as well.

The southern part of this tract adjoining the Conduit road is a wooded ridge, which separates the valley of Rock Run from the bottom lands of the Potomac river, which are here nearly half a mile wide. In this wooded district only a few shallow pits have been sunk for the purpose of tracing the croppings of some of the larger veins. The principal developments have been made in the open country beyond this ridge, on either side of the valley of Rock Run. As this valley runs across the strike of both country-rock and veins, about 100 feet below the top of the adjoining ridges, it affords an exceptionally large number of actual rock-outcrops. By means of these outcrops, by prospecting-pits, and by actual shafts and underground workings, there have been traced, with varying degrees of detail and accuracy, eight apparently distinct and parallel veins crossing the property from north to south, at distances from a few hundred feet to a few hundred yards apart. These I will describe by numbers, commencing at the west.

No. 1.—This vein, on the extreme western edge of the property, shows on the surface only siliceous schist and iron-stained, honeycombed quartz, like that described on the Ford property.

No. 2. *Pine Hill Vein.*—A pit has been sunk 30 feet on it, and a mill run of a 10-ton lot of ore yielded $8 per ton. It has been prospected over a length of about half a mile. It is quite wide, and consists of a series of thin quartz-bodies radiating upwards, separated by sheets of altered country-rock. It is supposed to be the same vein which, further north, is opened near the Eagle mill: here it is also very wide, and the gold, which is coarse, is mostly found on the two walls. Thirty to forty tons are said to have been crushed from it in this mill, with an average yield of about $10 per ton.

No. 3. *Broad Rock Vein.*—Has a cropping some 10 feet wide of quartz and siliceous schist, at the upper end of a little cañon where the hills come close to the stream of Rock Run. It has been traced southward to the top of the wooded ridge, and northward is supposed to run into the Eagle property. Assays of material from the cropping on the creek gave $13.13 per ton for the honey-combed slate rock on the eastern wall, $4.16 per ton for the schist carrying fresh white pyrite, and $4.17 per ton for pyritiferous quartz.

No. 4.—Crosses the creek near the lower end of the cañon or nar-

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rows. It has been stripped for some 20 feet on the north side of the creek, where it shows white quartz a foot or more wide, but in the creek-bed it is only marked by thin stringers of quartz in the schist.

No. 5.—Forms a jutting ridge on the south side of the creek at the lower end of the narrows, and is opened by a cut some 15 feet deep on the north side near the mill. In this cut the surface-soil is discolored by iron-stains, and the quartz only shows in the lower part of the cut.

No. 6. *Eagle Vein.*—Is opened by a 50-foot shaft on the Eagle property, north of the Harrison line. It is proved by various open cuts on the former property, and on the Harrison property west of the barn. Outcrops on the bluffs east of the Harrison mill, and in the hill south of Rock Run, are supposed to be on this vein. A mill-run of decomposed rock and soil, with some quartz, from the pit near barn, yielded $8 per ton, and of quartz from the brow of hill overlooking Rock Run gave $11 per ton.

No. 7. The Harrison vein, which showed very little cropping on the surface, has been most thoroughly prospected by pits 25 to 40 feet deep along the ridge on which the farm-house stands, and is now opened by a shaft 160 feet deep, from which levels or drifts have been run on the vein at depths of 90 feet and 160 feet respectively. The first level has been run about 300 feet, and the second about 200 feet, on the vein, which varies from 1 to 3 feet in thickness. No stoping has yet been done, but the mill, when running, has been employed upon material taken from the shaft and drifts, besides the various test-runs of other veins. The ore is not weighed, but its amount is estimated by the number of car-loads which are run on a tramway from the shaft to the mill. As I learn from Dr. Kempster, the highest mill-run after a clean-up has been $27.50. The lowest, representing a run of 17 days, was $10.42. This does not take into account, of course, what is lost in the tailings. The total amount of bullion produced by the mill to date is about $12,000.

No. 8 *Vein.*—This is a prominent outcrop of pure white quartz, a short distance south and east of the Harrison shaft-house, which shows no free gold, and scarcely an iron-stain. A mill-run of 1 ton of it yielded $7.50.

The Irma vein, about an eighth of a mile south of this, on the south side of Rock Run and on the north slope of the wooded ridge, appears to be on the same line of strike with this vein, viz.: from 5° to 10° west of north. It has been developed by a 50-foot shaft and about 100 feet of drifts, the ore from which is said to have

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yielded about $10 a ton in the 10-stamp mill on the property, which probably has considerable loss in the tailings.

This part of the vein is more irregular both in strike and dip than most of those thus far developed, varying from a strike of N. 60° E. with a dip of 50° to 60° S.E. at the north end, to N. 4° W., and a nearly vertical dip at the south end, which is about the normal direction. At this end there is no well-defined quartz-body, only a mineralized fracture or fault-plane in the decomposed country-rock.

About half a mile still further to the eastward on an adjoining property, known as the Lynch place, another quartz-vein has been opened by a shaft and considerable drifting, but is now closed down on account of some disagreement among the owners, though from all that can be learned the ore ran well in gold, and yielded, like most of these veins, fine specimens of free gold in quartz.

Another gold-bearing quartz-vein has been developed at quite a distance from this group of properties. This is on the Huddleston farm, on the east fork of Cabin John Run, an important tributary of the Potomac, which takes its rise near Rockville, the county seat. It is about 5 miles in a straight line, a little north of east, from the Harrison mine, and 8 miles northwest of Washington. It is opened by a 50-foot shaft, which is connected by a tunnel, at 36 feet, with the bottom of the neighboring stream, and the vein has been drifted on for a short distance. The shaft was closed at the time of my visit. Surface-pits have disclosed the croppings of the vein several hundred yards to the northward. Its direction is about north, magnetic, and it is said to be from 2 to 4 feet wide. It has yielded some very rich and beautiful specimens of free gold.

Croppings of other veins have been prospected on the farm adjoining this on the south, and also near the Little Falls of the Potomac about half a mile back of the River View Hotel, just above the Conduit Road, which lies a little east of south from the Huddleston vein.

A belt, some 7 or 8 miles wide, is thus roughly defined as gold-bearing, though, as may readily be perceived from what has gone before, it is too early to say much about the distribution of gold in the region. Considerable prospecting has undoubtedly been done; but most of the people have little familiarity with mining, or the manner of occurrence of ores. Prospecting is, moreover, much more laborious and expensive in a region which is covered to so great a depth by decomposed and altered material than it is in the Rocky

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Mountains or in the Northern States, where the relatively undecomposed rocks are laid bare.

It is only in the unaltered rock, and by the inspection of extensive underground workings, that one can form any correct or definite idea of the geological relations of an ore-deposit. Such conditions are found only in the second level of the Harrison mine, and the extent of this is, as yet, so limited, that any generalizations that may be made from the conditions that prevail there are necessarily of a very tentative and preliminary nature, and liable to be modified when more extensive workings have been opened.

Country-Rock.—The country-rock from the lower levels of the Harrison mine is a greenish striped schist, containing both chlorite and epidote as basic silicates. These minerals, as is well known, are generally of secondary formation in rocks; hence the freshest-appearing rock is an alteration-product of some earlier one, it may be of one similar to the micaceous schist exposed in the lower cañon of the Potomac. Whether both were originally of sedimentary origin it is yet too early to give a definite opinion, though it seems very probable that when they are traced on the strike into a less metamorphic condition this may be found to be the case. As no considerable cross-cutting has yet been done in depth, other varieties of schist have not been distinguished; but it is probable that several exist, from the following observations: In exploration-pits on the wooded ridge, the country-rock is found to be altered in one place into a white powdery mass, which contains a large proportion of hydrated aluminous silicate or kaolin, while another, some distance to the eastward, shows a true soapstone, a hydrated magnesian silicate, which must come from the alteration of an essentially different rock from the former. The soapstone seems to occur along fracture- or fault-planes, and corresponds in chemical composition to those described by Mr. P. H. Mell,* from the Southern States, in a similar belt of rocks.

The following analyses of the above materials were made by Mr. L. G. Eakins, in the laboratory of the U. S. Geological Survey, No. 1 being the minute crystals constituting the greater part of the white powder, and No. 2, the soapstone.

The lamination of all these rocks has a very regular and persistent strike, a few degrees east of north, and a dip slightly to the east of vertical. It may be parallel to the original bedding-planes of the rock; but these are now difficult to distinguish and may have been obliterated by metamorphism.

* *Trans.*, x., 318.

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	1.	2.
SiO_2	48.17	58.82
Al_2O_3	38.06	7.82
Fe_2O_3	.....	
CaO	.....	Tr.
MgO	.....	29.68
H_2O	11.15	4.52
	97.38	100.84

Quartz-Veins.—The quartz-veins have also a very persistent strike a little west of north—generally N. 9° W.—and dip in depth a little to the west of vertical. It is noticeable that in the upper part they generally dip slightly to the eastward; in other words the tops seem almost always to turn over to the westward, as if there had been a general movement of the upper portion of the rotted rock in that direction. This can hardly be explained on Prof. Kerr's theory of frost, for it extends too deep for frost to have reached in this region. Its general occurrence in the region is remarkable. If it were generally observed over the whole country, one would be apt to notice that the apparent movement of the rotted surface is in the opposite direction to the rotation of the earth.

While so regular in their larger features the veins are yet quite irregular or wavy in detail; that is they often bend in and out, both in dip and strike, within short distances. They are liable to wedge out altogether, leaving only a mineralized zone to mark the fracture plane, with occasional thin sheets of quartz within the foliation of the rock. They undoubtedly follow fracture- or fault-planes. In the lower workings of the Harrison mine, the parallel banding, striated and polished surfaces, and some friction-breccia can be distinguished, and bands of altered schist are often included in the quartz—all of which is characteristic of fracture-planes; and yet these phenomena are much less distinctly marked than in the Rocky Mountains. A band of comparatively unaltered pyrites generally accompanies the vein. These are of the lighter variety, generally known to western miners as white iron; they carry gold, but do not seem to be as rich as the more easily decomposed pyrites.

While the fracture-planes can be traced with great regularity for considerable distances, the quartz appears to vary rapidly in thickness from point to point, sometimes being five feet or more in thickness, more generally one to two feet, and then disappearing altogether, to

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be taken up further on, along the same or an adjoining fracture-plane.

This irregularity renders it difficult to estimate the amount of quartz which may be obtained from a given vein; but the abundance of the veins in a given area would seem to make up for the limited width of single veins.

Occurrence of the Gold.—The gold is found in the pure white quartz free from pyrites; associated with the pyrites; and in the pyrites themselves. Dr. Kempster has some remarkably interesting specimens in which free gold can be detected in the interior of well-defined crystals of pyrites. It is not, however, confined to the quartz, but is found in the country-rock adjoining it on either side; and these bands of sheeted, decomposed country-rock are often richer than the vein itself.

Besides the gold and pyrites, galena and some silver are found irregularly distributed along the veins. From a property just north of the Harrison, a steel-gray mineral has been obtained which, from examination with the blow-pipe, Dr. Kempster considers to be tetradymite or telluride of bismuth. This is not at all unlikely, as this mineral has been found in several mines in Virginia, and also in North Carolina and Georgia. The association is particularly interesting at this time in view of the interesting remarks of our President, Mr. Pearce, upon the association of tellurium and bismuth with gold that has come under his observation in Colorado mines.

The gold obtained from these mines is unusually pure, running from 840 to 960, and even higher.

General Conclusions.—With the very meager data derived from personal observation of this portion of the auriferous belt, and the extremely uncertain and contradictory information which is obtainable with regard to the southern portion, it is impossible to form any very decided opinions as to the future of these deposits.

In endeavoring to form an opinion as to the prospective value of a newly opened mining region, one naturally lays much weight upon the previous history of regions already worked under geological conditions most nearly resembling it. The geological conditions of this region are apparently very similar to those of the entire auriferous belt of the Appalachians, extending from here southward to the mountainous portions of Georgia and Alabama, and most nearly resemble those of the middle portion of Virginia. In all this belt there is no record of any great mine the product of which can compare with the few enormously productive mines that have started the

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successful mining districts of the west; and there seems to have been a very large proportion of disastrous failures among the many gold mining enterprises that have been undertaken here. But, as I have already remarked, there is some reason to assume that many of these failures have been due to ignorance and bad methods of working. The depth to which the rocks have been rotted and decomposed has, moreover, been a serious obstacle in the way of arriving at a proper understanding of the geological relations of the deposits, and while it has tended to make the surface-showing unduly rich, has undoubtedly been an important factor in preventing systematic and successful mining in depth. Some go so far as to say that gold in this belt ceases in depth beyond the water-level, or zone of surface-decomposition of country-rock. For such a statement I can see no valid geological reason. That the chemical composition of ores changes below this line is recognized everywhere and is readily explained; it is, moreover, an observed fact, that is not always susceptible of scientific explanation, that the metallic bases contained in an ore in some cases undergo a remarkable change in depth.

It would naturally be expected in a region where decomposition has gone on uninterruptedly for so long a period as it has in this belt, that by the gradual removal of the more readily attackable and less valuable constituents, the proportion of gold (a metal that is practically uninfluenced by the agents of decomposition), remaining in the decomposed and superficial portion of the region, should be greater than it was in the original deposits before decomposition had set in. Hence the deposits will contain on the average somewhat less gold to the ton from the surface downwards until the unaltered rock is reached, and in this rock they will vary in richness from point to point as do all ore-deposits, but any assumption of a disappearance of the gold in this unaltered rock is geologically untenable.

After the water-level is passed the sulphurets will be found in an unoxidized condition, and such gold as is actually contained in them cannot be separated by the ordinary process of amalgamation. It can, however, be obtained by suitably conducted chlorination or by smelting. To how great a depth it will be profitable to work a given deposit is then a question not for the geologist but for the technical mining engineer. It may be that, while the deposit still carries gold in a proportionate amount to what it did above, it will be no longer profitable to work it, on account of the increased cost of mining and reduction. The plant that successfully reduced the free gold-ore at the surface may require considerable and expensive modifications to

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adapt it for the treatment of unoxidized ores. Although this treatment presents many practical difficulties, and has in a great number of cases proved commercially unsuccessful, it is based on well-known chemical and metallurgical principles and involves no hitherto undiscovered chemical processes. The question is simply, whether the expense of treatment will be greater than the average yield of ore, and is one that a competent mining engineer should be able to answer by practical experiments before establishing an expensive plant. Of those patent processes by which it is claimed gold is extracted from the sulphurets by a secret and hitherto unknown chemical reaction it may be safely said, that either they will not do what they claim when applied on a large scale, or that they contain nothing not already known to the science of ore-reduction, and hence nothing worth paying a royalty on.

The most important question from a geological standpoint, with regard to these deposits, is, whether the quartz-bodies have been formed along fracture- or fault-planes, or are simply lenticular masses following the bedding of the schists. The latter view has been held by many in regard to the auriferous belt in general; but what I have seen of the Montgomery County deposits leads me to regard them as of the former class, and hence belonging to the category of true veins.

Judging from the developments thus far made, and from the great number of parallel veins in a comparatively small area, it would appear that these deposits have been made along a series of co-ordinated fractures or "compression joints," * in which the predominating direction is nearly north and south, with comparatively few cross-joints, or fractures at a considerable angle to this direction, such as are common in a region where a torsional strain has been associated with that resulting from compression. It is probable, therefore, not that any very great single veins, of extent and width comparable to those found on the more prominent "fault-fissures" of the west, exist here, but that the vein materials are distributed along a series of small co-ordinated fissures, often so nearly on a line as to appear to be part of the same vein, when traced by isolated outcrops alone.

It is likely, also, that in a given vein the quartz will be found to be very irregularly distributed, widening rapidly at one point, thinning out again in comparatively short distances, often almost en-

* *Trans.*, xvi, 824.

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tirely disappearing, even though the fracture-plane can still be distinguished and gold-bearing material be found along it. But even after it has disappeared it is likely to come in again, if not on absolutely the same plane, on one nearly adjoining and parallel with it. The underground workings are still too limited in extent to afford any data for estimating the average extent of these quartz-bodies, but it is probable that, when sufficient mining has been done, it will be found that there is some sort of system in their distribution, as there generally is found to be in the distribution of bonanzas or rich ore-bodies in the greater veins. A few fundamental principles may be laid down which should govern the development of new districts of this nature.

It should be borne in mind that the quartz is not the only vein-material; it was probably deposited more or less contemporaneously with the gold, pyrite and what other metallic minerals may be found. The quartz filled whatever openings were formed by the fracturing and faulting of the country-rock, but the solutions that accompanied it penetrated the fractured and sheeted country-rock on either side, depositing there pyrite and gold with some silica, so that a portion of the country-rock constitutes ore as well as the quartz. How much this may be in the case of a given vein can only be determined by actual test, and it behooves those working the mines to have frequent tests made of the material along the sides of their vein, either by assay, or if this is not practicable, by panning, to determine the width of vein-material which it will pay to reduce. It will be found to vary very much from place to place; and it is evident that profitable results cannot reasonably be expected, if good ore is thrown away, or material which costs more to reduce than it carries in gold is, to any considerable extent, passed through the mill.

In such a district as this, it is of especial importance that underground exploration should be carried on far in advance of actual reduction of ore. No extensive reduction-plant should be erected until an amount of pay-ore sufficient to supply it for several years has been actually opened and fully tested. Underground developments should not be confined to a single vein, but frequent cross-cuts should be made in the softer and easily-worked ground immediately above the hard rock, to determine the existence of parallel veins to be opened; for it is only by working great quantities of ore that costs of reduction can be reduced to a minimum, and large profits be obtained; and such large quantities can hardly be expected

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from a single vein. The number of the veins gives reasonable promise for such production, however, if mining is conducted systematically; and the mine-owner here has a great advantage over his Western brother, in that he is not exposed to annoyances resulting from the mining law of the United States, but has a clear title to all the ground within the vertical side-lines of his property, and, when he has developed a valuable body of ore underground, runs no danger of having it taken away from him by his neighbor, who, after such discovery, may have constructed an apex for the body within his own side-lines.

A separate and distinct problem is presented by the evidently large amount of gold contained in the soil and loose material near the surface along the croppings of the veins. Such material can evidently be treated very cheaply, if its working is conducted systematically, and on a sufficiently large scale. It must first, however, be carefully prospected and tested, so that the amount available and its average yield can be determined with reasonable accuracy. The most successful treatment of low-grade gold-ore is that of the Homestake mine in the Black Hills, which yields an average of only $3.71 per ton in gold and silver, but which is mined and reduced at a cost of $2.52,* leaving a profit of $1.19 per ton. It is to be noted, however, that this low cost of treatment, and the obtaining of a profit from such ore, is only possible, because nearly one-quarter of a million tons per annum are treated, and that similar deposits in that region, worked in a small way, have not proved profitable.

To work this material at a profit, operations must also be conducted on a large scale, for the average yield will probably be low, though the gold is in a most favorable condition for treatment. In Georgia, similar material has been treated by turning a head of water on the ground and washing it down in boxes to stamp-mills. No large streams are available here which would give such a head, and hence some method of handling, other than by water, and consequently more expensive, must be used. It is probable, however, that this material can be pulverized for amalgamation by other machinery than by the power-wasting stamp-mill, and more cheaply, provided a machine be selected of sufficiently simple construction and effectiveness of power. Most of the machines which are advertised to do such work, are too complicated in construction for successful use by the ordinary mill-hand, and do not present a thoroughly effective use of power from a mechanical point of view.

* Carpenter, *Trans.*, xvii, 570.

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ELECTRICITY AND HAULAGE.

BY FRANCIS A. POCOCK, M. E., SCRANTON, PA.

(Washington Meeting, February, 1890.)

THE writer is continually asked, if this electricity is all you claim for it, why do not the mines put it in and use it? The best answer was given by Mr. John Fox Tallis, in his paper read before the South Wales Institute of Engineers in 1888, namely, that the reasons are similar to those which have prevented the more general employment of other systems of haulage. Managers, as a rule, are "not experienced mechanical engineers; neither are they all versed in electrical engineering." It may be said that there are a number of "able and practical electrical engineers who could do the work; but those electrical engineers are not conversant with the practical routine of colliery work and are, therefore, placed at a disadvantage by not knowing the actual requirements of mining engineers, and the many little difficulties to be met and contended with in underground workings; and, until mining engineers have acquired the necessary knowledge and a confidence in electricity, it is only natural that they will continue to follow the beaten path."

In the above remarks, Mr. Tallis has hit the nail on the head. In the face of the partial failure, financially, of the haulage-plant which the Union Electric Company of Philadelphia had set to work at Lykens Valley, Pa., an electric haulage-plant was installed near Scranton, Pa. The question was not whether electricity could do the work, but whether any electrical company knew enough about the practical difficulties to attack and overcome them. That point being settled, the remainder was easy.

The contract for the Erie Colliery haulage-plant was signed in January, 1889.

This colliery is owned by the Hillside Coal and Iron Company, Mr. Samuel Hines, President, and Captain W. A. May, Superintendent. The company has been using arc-lamps in its breakers for three years. It is the only company in the Lackawanna Valley which has had enterprise enough to do so. Consequently, it was acquainted

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with electricity before being asked to look upon it as a power, and, therefore, was more easily persuaded to see in it something more than a toy. It had saved the company money, and the company was ready to give it a chance to save more.

Nearly three years ago, the Erie breaker was burned down. This led to the introduction of the arc-light. The new breaker was built with a capacity of 700 mine-cars per day of ten hours. The old sidings at the foot of the shaft had not room for more than 15 cars. The shaft had to wait for the cars, and the breaker for the shaft. The average output was 510 cars per day. It was dependent upon an inclined plane on one side and a slope on the other, neither of which was capable of supplying the shaft alone. The question presented to us was, Is it possible to arrange sidings so as to make the shaft independent of slope and plane for half an hour or more?

After careful investigation, levellings and surveys having been made, I answered in the affirmative. This meant more standing-room at the shaft-foot. We wanted 40 loaded and 40 empty cars or more. The grades were against this. The pillars were too small to cut. By running a road through old workings, we got round the shaft. One-third of the coal would have to be backed into the sidings. Mules could not do the work. Rope-haulage would have been too complicated. It was decided to drive the road suggested, and give electricity a trial. About the middle of May this road was ready; but no track had been put down. We had standing-room for 70 loaded cars and 50 empties, making more than an hour's run. The electric locomotive went to work on the 8th of October, having been run over the track a few times to test the position of the switches. These did not require much alteration, and the machine appeared to hold the track well and pass the curves easily. The mules held their own up to noon. Then the locomotive came on the scene; the mules and the drivers were side-tracked, and the "Buzzer," as the boys named the machine, went to work. For seven or eight days the mules and boys came out, ready to take hold; but it was of no use—the "Buzzer" appeared capable of looking after all the work.

After a time, the mules got tired of doing nothing and used to make most piteous noises, it is presumed, from cold. They were sent to work in other parts of the mine. The locomotive held its own, and gradually worked up the output. There was considerable talk about a certain record that the mine had once made, of 640 cars. About two weeks after starting, this record was beaten by hoisting

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330 cars before noon. Since then there has been nothing good enough for the "Buzzer." Again and again it has beaten its own record, and has often brought 375 cars out in time to hoist before noon. Six hundred cars is now a small day's work. The average daily hoisting for the first month was 667 cars.

Thus things went on, until it was thought there was no breakdown in the machine. But on the morning of the 3d of February, the trolley-arm was thrown by a bent insulator, torn from its place by a plank, and the "Buzzer" was quiet. The mules came in again, but could do nothing with the work. About 45 cars were all they had time to haul before they were made to stand out of the way. Mr. Wyman, the constructing engineer for the Electric Company, had rigged up a jury-trolley, and, after about two hours' stop, the machine was at work again, with a small boy to operate the make-shift trolley. The mules went home. On the 4th, work went on all day, with the boy and the make-shift. The 5th saw the trolley all right again.

The machine thus had one mishap and stopped working for two hours, out of four months. The owners give it credit for 80 to 90 tons per day more than mules could haul. They are so well satisfied that the 25-pound rails are being replaced by 40-pounders. The company is also now in negotiation for another plant for one of its soft-coal mines.

In describing the plant, it will be most convenient to commence with the prime mover, and work down.

Boilers.—No addition was made to these, there being sufficient capacity already.

Engine.—This is an Armington-Simms, of 9½ inches diameter and 12-inch stroke; 278 revolutions per minute. Steam-pipe, 3½ inches in diameter. The driving-wheels, of which one is used, have 60 inches diameter and 10½ inches face. At 70 pounds boiler-pressure, the engine gives 41 indicated horse-power. At 60 pounds, the ordinary pressure, only 34.3 horse-power is given off; but this has so far proved equal to the work.

Dynamo.—A 50 horse-power Thomson-Houston machine of 220 volts at the terminals, pulley 15 inches in diameter, speed 1150 revolutions per minute, is used.

The engine and dynamo stand in the hoisting-engine house with the other engines required for hoisting, running the breaker and pumping.

Instruments.—A volt-meter, am-meter and automatic cut-out are on the wall, while two lamps of 16 candle-power each give the man

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in charge a good idea of what is being done in the mine by the locomotive. The main feed-wire runs out of the engine-house into a 1½-inch gas-pipe, which protects the wire in the shaft. It is a No. 0 Clark wire (0.34 inch in diameter), 260 feet long to the switch at the foot of the shaft. The return-wire is bare copper, hung in the shaft without any protection at all. The total loss in these wires when running 50 horse-power is under 2 horse-power, or 4 per cent.

From a switch at the foot of the shaft, the current passes to the trolley-wire, which is of hard-drawn copper wire, without insulation, and is made fast by the following arrangement: A hole is drilled in the top-rock, and a wedge-shaped bolt, with a split gas-pipe around it is inserted into the hole. A piece of dry, well-painted wood, 4 by 3 inches in size and 18 inches long, has a hole bored through it near each end, and the bolt is put through one of these holes and the nut screwed up. Two such bolts make each block fast to the roof. The mine-insulator, made of cast-iron, is then screwed into the block, and the wire soldered into it. Where two lines of wire join, a frog, much the same as that in use on the surface roads, is inserted, and no difficulty is experienced in passing thirteen of these switches in every trip.

The foot of the shaft and the gangways along which the locomotive runs are lighted by 50 incandescent lamps of 16 candle-power each. These were put in as an experiment by the Electric Company and accepted, as they have proved of the greatest service. Electric signal-lamps are also used, which show the condition of the work away from the shaft, and also tell the engineer when the track is clear and all switches are set for him.

The motor used on the locomotive is a Type "G," of 40 horse-power capacity. The rheostat is controlled by hand from each end, thus allowing the man in charge to regulate both power and speed. The brake is in front of him, and the reversing-lever at his left hand. The length of the locomotive is 9 feet 7 inches, width 5 feet 4 inches, and height 5 feet 6 inches. Of this height, 16 inches are taken up by the rheostat. This will, however, be placed at the end of all future machines, so that the total height of this class of locomotives will be 4 feet 4 inches. The gauge is 36 inches, and the weight was originally 10,500 pounds; but, the mine being quite wet, this was increased, for the sake of traction, by 1800 pounds, so that the running weight is 12,300 pounds. The largest load that has been hauled by this machine is 33 cars, weighing about 6300 pounds each, or 207,900 pounds—nearly 104 net tons, or about seventeen

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times its own weight. This was started on a down-grade of about 1 per cent., and shoved along a nearly level road to the shaft, the whole being on curves. It is rarely that the engineer can see the light carried by the trip-rider so continual are the curves in this mine. The locomotive is run by one man, with a boy who makes up the train and turns the switches. It displaces seven mules and three drivers, leaving no margin to reduce horse-power expenses. But as this was not expected when the plant was contracted for, no disappointment has been experienced, the plant having been put in to get more coal to the shaft than mules could haul.

The locomotive has demonstrated that, complying with the anthracite mine-law of not running more than 6 miles per hour, it can hold the following schedule:

East or Slope Side.

Distance run per trip, including making up, etc.,	2884 feet.
Time of trip,	10½ minutes.
Cars per trip,	15.
Trips per day,	16.
Miles run per day,	8.73.
Total time,	2 hrs. 40 min.

Locomotive reversed 128 times per day.

West or Plane Side.

Distance run per trip, including making up, etc.,	2546 feet.
Time of trip,	6½ minutes.
Cars per trip,	25.
Trips per day,	26.
Miles run per day,	12.55.
Total time,	2 hrs. 50 min.

Locomotive reversed 104 times per day.

Total cars, 890; total time running, 5 hours 30 minutes; total reversals, 232; total miles run per day, 21.28.

Time occupied on run 6 miles per hour,	3 hrs. 33 min.
Time occupied in reversing,	1 hr. 57 min.
Time available for contingencies,	4 hrs. 30 min.
Total time,	10 hrs.

As it is impossible to get a full trip every time, it will be seen that there is fully time to get the 700 cars for which provision was made, and which the breaker is capable of treating.

The principal difficulty in the Erie colliery for any haulage-system arises from the mine-gangways round the shaft having been driven on water-level. This made everything far from straight.

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The pillars near the shaft were too small to cut, so as to get a straight road without drowning it out, by going below the water-level. The east or slope side might have been run by gravity, if the cars had been better, but as they were the grade could not move them. The west side was too crooked to get a rope to run well. The present arrangement of tracks requires all the loaded cars to be shoved into the siding, and the empty cars on the east side to be shoved up a grade of 3 per cent.

All of this shoving is done on curves so that the cars only get the pressure on one bumper. The bumpers are on the sides of each end, which makes the work harder than if they had been in the middle.

There are two gauges of cars, the 28-inch going to the east side, and the 36-inch gauge to the west. The switching is apparently quite complicated, and I am not aware of any other system which can satisfactorily accomplish this.

The single overhead wire gives itself readily to this complication, and has, I think, other material advantages, not the least being the extra traction. Experiments on the Reading and other railroads prove that some 15 per cent. more traction can be obtained by passing a current from the engine to the rails. Again, there is less wire overhead. Under certain conditions a double overhead wire might be, but, ordinarily, the single is preferable.

It is to be regretted that the mine has worked so few days (only about half time) since the machinery started; but the test has been thorough. After standing eleven days, the rails were red with rust, giving a high resistance to the current, the cars all stiff with standing, and the overhead conductor covered with wet dust; yet the locomotive gave a thoroughly satisfactory account of itself.

So much has been said about the "impracticability" of overhead wires and applying electricity in the shape of a locomotive, before the Mining Institutes of Great Britain, that it is very satisfactory to place something on record regarding the actual work done by this condemned-before-tried application. I do not wish to say that the locomotive is the best way to use electricity for haulage; but I do say that, under some conditions, it is the easiest way to overcome certain difficulties. What is the best system to be employed must be settled by the conditions at the particular place. It is, therefore, far from wise to attempt to decide *a priori* what cannot be done. Most things are at least possible. As a general rule, it will be necessary to go into the workings and examine every detail of the work before giving any opinion.

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Electricity offers a very agreeable solution to many mining problems. That a broken incandescent lamp may fire gas has been placed beyond doubt by the experiments of Mr. Morley and Lieutenant Hutchins, as is probably known to most of our members; but the wide range of duties which will not take the current into danger is open to all, and in electrical mining appliances there is a new field for the inventor, which ought to be as profitable as any that have gone before it.

The question of cost is ever foremost. To throw a little light on this subject, I reproduce a table published by the *Colliery Engineer*, which appears to be a fair approximation to the actual cost of electrical plants for general mining work.

Distance Transmitted in Feet.	Electro-Motive Force at Terminals of Motor. Volts.	Commercial efficiency. Per cent.	H. P. required at Generator.	Cost of Generator, etc., per H. P.	Cost of Motor, etc., per H. P.	Total Cost of Electrical Equipment, Including Wire, per H. P.
5,000	400	65	1.53	$61.20	$45.00	$123.42
10,000	400	54	1.85	74.00	45.00	155.08
15,000	400	47	2.13	85.20	45.00	182.00
5,000	800	71	1.41	59.93	47.50	117.35
10,000	800	65	1.53	65.03	47.50	130.46
15,000	800	59	1.69	71.83	47.50	147.06
20,000	800	54	1.85	78.63	47.50	163.71
10,000	1,200	69	1.45	65.25	50.00	128.21
15,000	1,200	65	1.53	68.85	50.00	137.50
20,000	1,200	61	1.64	73.80	50.00	149.12

The figures are for units of from 20 horse-power up. They do not include the cost of the engine and steam-plant to run the generator.

Mr. Wm. Geipel, of Edinburgh, also gives some interesting figures of the first cost of plant per horse-power transmitted in England.

Total Power Transmitted.	System of Transmission.	Distance of Transmission in Yards.			
		100.	1100.	11,000.	22,000.
		£	£	£	£
5 H. P.	Electric	75	31	142	210
	Hydraulic	41	97	610	1880
	Pneumatic	73	210	1090	2060
	Wire Rope	6.5	61	760	1220
100 H. P.	Electric	32	35	59	87
	Hydraulic	14	28	164	310
	Pneumatic	26	34	109	192
	Wire Rope	1.1	8.4	81	162

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Also, to obtain an idea of the comparative cost of installing and running tail-rope, electrical locomotive and mule-haulage plants, I have taken a road worked by mules, where the actual output is about 1000 tons of 2000 pounds per day, and have estimated upon the electrical and tail-rope plants, to haul the capacity of the shaft and breaker, about 1400 tons per day.

In each case the haul is 6000 feet long, the day ten hours. Cars weigh 2000 pounds empty, and carry 4000 pounds load, making 6000 pounds total weight. The traction on the level is about 70 pounds per ton.

First Cost of Mule-Haulage.

16 mules large, valued at $175.00 each . .	$2800.00	
16 sets of harness at $20.00, ...	320.00	
Total cost, ...	$3120.00	

Interest, Depreciation, etc.

20 per cent. depreciation, on $2800, . .	$560.00	
6 per cent. interest, ...	168.00	$728.00

Working Expenses for Two Hundred and Sixty Days.

16 mules, harness, shoeing, feed, care, at 30 cts.,	$1752.00	
8 drivers, at $1.75 per day, ...	3640.00	$5392.00
Total yearly expense, ...		$6120.00

Or 2.354 cents per ton for 260,000 tons.

First Cost of Tail-Rope Haulage-Plant.

Engine, pair of 14-inch by 20-inch, 150 rev. per minute, .	$2000.00
18,000 feet of 1-inch steel rope, ...	3456.00
150 rollers, 8½-inch by 7¾, ...	786.00
100 sheaves, 2 feet diameter, and one 6-foot sheave, . .	1700.00
Cost of erection, etc., ...	1000.00
Boilers, ...	1500.00
Total first cost, ...	$10,442.00

Interest, Depreciation, etc.

6 per cent. on $10,442, ...	$626.52	
Engine, depreciation 10 per cent., repairs, 5 per cent., ...	300.00	
Boilers, depreciation 10 per cent., repairs, 5 per cent., ...	225.00	
Ropes, life 1 year, ...	3456.00	
Sheaves, life 2 years, ...	850.00	
Rollers, life 1 year, ...	786.00	$6243.52

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Running Expenses.

Engineer at $50 per month,	$600.00	
Boiler-attendance,	300.00	
Trip-rider, 260 days, at $1.75	455.00	
Oil and waste,	300.00	$1655.00
Total yearly expense,		$7898.52

Or, 2.169 cents per ton for 364,000 tons.

Note.—Foundations and steam-pipe have been omitted.

First Cost of Electrical Haulage-Plant.

For 1400 tons of 2000 pounds, hauled 6000 feet per day of 10 hours, cars same as rope haulage:

Engine, generator and motor (locomotive),	$7500.00	
Boiler (all considered as moving parts),	1000.00	$8500.00
Wire, insulators, switches, etc.,	1300.00	
Labor in erection (all standing parts),	1200.00	2500.00
Total first cost,		$11,000.00

Interest, Depreciation, etc.

Interest, 6 per cent. on $11,000,	$660.00	
Depreciation on moving parts, 10 per cent.,	850.00	
Repairs on moving parts, 8 per cent.,	680.00	
Depreciation, standing parts, 6 per cent.,	150.00	
Repairs, standing parts, 6 per cent.,	150.00	$2490.00

Running Expenses.

Engineer on surface at $50 per month,	$600.00	
Engineer on motor at $2 per day,	520.00	
Trip-rider at $1.35 per day,	351.00	
Oil and waste,	150.00	
Sand,	100.00	$1721.00
Total yearly expense,		$4211.00

Or, 1.15 cents per ton for 364,000 tons.

Note.—Foundations have been omitted.

Many of the advantages of the electrical system over a mule- or rope-method of operation are obvious to the superficial observer. I may mention the absence of soot and moisture, the former owing to the oil-lamps ordinarily used, the presence of which can be de-

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tected at some distance. As to the latter point, the presence of animal exhalations from mules in mines is conspicuous by its absence immediately on the introduction of an electric system. In this and other ways the usual foulness of the air of the mine is greatly remedied.

The electric lights, which may be run off the same wires as furnish power for the locomotive, are not only a matter of convenience, but add much to the safety, and, according to prominent engineers, to the economy of working the mine. The men may exert their full strength without fear of accident; and, moreover, the saving of time now occupied in trimming and attending to lamps is an item worthy of consideration.

After the introduction of an electric locomotive, it may be found advisable to use a stationary motor for inclined hoisting, to haul the cars up steep grades to the main line.

For pumping, the system is very conveniently applicable, as the pumping units may be located at any desirable points, and of such power as the occasion may demand.

One mine superintendent, who is thoroughly conversant with the subject, recently declared that, laying aside all question of financial saving, and on humane principles only, he would recommend the system as doing away with the abuse of mules by ignorant, cruel men and boys, in whose charge the animals are placed.

Numerous other economical and convenient applications of the electric current will be found in mine-operation, both above and below ground, which need not be here mentioned. From the standpoint of safety and general utility, science seems to have produced as yet nothing more worthy of careful consideration.

For the benefit of those who are interested in electric power-transmission the following table has been compiled, which shows the reduction in the cost of wire in transmitting the current by raising the voltage in the mines.

The loss in the generators in this table is assumed at 10 per cent., which is higher than the guarantees of first class companies, and the loss in the motors at 10 per cent., making a loss of 19 per cent., to which the loss in the wires must be added, so that over a distance of 2000 feet and return, with complete metallic circuit and 2000 volts potential, the loss would be 19 per cent. in motor and generator, and 10 per cent. in the wire, or 29 per cent. in all; whereas, with 1000 volts, the losses would be 19 per cent. and 2.77 per cent., or 21.77 per cent. The above are approximations only, and the losses must be calculated for each case.

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Voltage at the Generator.	Total Current in Amperes.	Reistance of Conductor in Ohms.	Diameter of Wire in Inches.	Weight of Wire in Pounds.	Cost of Conductor at 17 cents per pound.	Loss in Conductor.	
						H.-P.	Per cent.
200	273	0.09	0.7136	6166.8	$1,048.35	9	10
400	137	0.2272	0.436	2312.48	393.12	5.71	6.34
500	109	0.3412	0.356	1541.68	262.08	5.43	6.03
600	91	0.3788	0.3385	1387.48	234.87	4.2	4.66
800	68	0.487	0.2985	1079.2	183.56	3.01	3.34
1,000	55	0.620	0.2646	847.9	144.14	2.51	2.77

The table shows the reduction in first cost by using a high electromotive force, thus allowing the utilization of water-power or other cheap generating methods within a reasonable distance of the location of the motors.

For this purpose a current of high potential may be used for transmitting the energy to the shaft of the mine, the current being there transformed into so low a tension as to present absolutely no danger to those who may accidentally come in contact with the wires or apparatus.

While a current of 500 volts, as adopted for a standard in street-railway work, and in general for power-purposes above ground, is not dangerous to life, or capable of producing serious injury, it has been deemed advisable by some of the leading electrical companies to adopt 220 volts as a standard for underground work, since this current may be taken with impunity by any one. One peculiar fact in this connection is that horses and mules are seriously affected by a current which may be taken almost without notice by men; and it may not be amiss to remark that a human death has never yet been caused by a current of 500 volts potential, sensational accounts in some newspapers to the contrary notwithstanding.

The following account of a most interesting test-experiment was given in the paper of Mr. A. T. Snell, read before the National Association of Colliery Managers at Wigan, England.

Normanton Pumping and Hauling Plants.—I will first refer to a small pumping plant, erected at St. John's Colliery, Normanton, in August, 1887. Mr. Warrington, of Messrs. Locke & Co., has kindly allowed me to give the following details: The pump delivered about 39 gallons per minute through 530 feet head. A very complete test of this plant was published in the mining journals about November 18th of the same year. I have given the figures in tabulated form for comparison with tests of later plants. The results obtained were so satisfactory that Messrs. Locke

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& Co. decided to put down a larger plant, to deliver 120 gallons per minute through 900 feet of vertical head, and I was intrusted with the designs of the machines. This larger plant was started in February of the following year, 1888, and has continued running about 22 hours per day since then. The run now exceeds 18 months, and during that period there have been no breakdowns traceable to the electrical details, all stoppages being due to mechanical defects, incidental generally to the engine and pumps. The plant has been described before, but not in detail. I now propose to give results of tests made by the colliery engineer, Mr E. Brown, and myself. The engine is of the semi-fixed type by Robey & Co., of Lincoln. It is compound and is rated by the makers at 30 nominal horse-power. It has indicated during the past 18 months on an average about 80 horse-power. The dynamo is driven off the fly-wheel by a link belt 14 inches wide, curved to fit the pulley. The belt speed is about 2750 feet per minute. The dynamo is designed to give 600 volts and 70 amperes. The motor is of similar design to the dynamo, but the field-magnets are of lighter construction. The pumps are differential, with two 6-inch and 4½-inch rams. The suction is made by the two large rams only. On the in-stroke the 6-inch rams deliver water partly into the rising main and partly into the small rams. On the out or suction-stroke the 4½-inch rams deliver into the column and so the discharge is fairly continuous. When doing full duty, the pumps make 25 revolutions per minute. The rising main is about 450 yards long, and is composed of 4-inch cast-iron pipes. An air-vessel about 5 feet high is fitted at the lower end near the delivery-clacks. The piping is undoubtedly too small for 120 gallons per minute. It was really designed for a feeder of about 50 gallons. The water travels in the column at about 250 feet per minute, and there is nearly 10 horse-power lost in friction. This heavy loss is against the total efficiency of the plant, and an allowance can be made if any one thinks proper. The cable is built of 19 strands of No. 17 B. W. G. copper wire, insulated and lead-covered. It is about 1000 yards long, and has a resistance of about .5 of an ohm.

About one week after the erection in February, 1887, the electrical quantities for the full load averaged about 65 amperes and 603 volts, with 450 revolutions of the dynamo and 134 revolutions of the engine per minute. The motor ran at about 450 revolutions per minute, and the pumps at 25 revolutions. Under the above conditions the water delivered was measured and found to be between 118 and 120 gallons per minute. The suction has a rise of about 14 feet, and the column is 860 feet in vertical height. Thus the theoretical horse-power in the water is about 32. The engine was indicated at the same time, and found to be giving 80 horse-power. The efficiency of the system, that is, the ratio between theoretical work in the water and indicated horse-power of engine, was therefore equal to $\frac{32}{80} = 40$ per cent. These tests have been repeated from time to time by various gentlemen, and the readings show a gradual decrease of the loeses due to friction, particularly in the engine and pumps. A log has been kept, and the daily readings of current show a steady fall. At the last test, Mr. E. Brown found the current had fallen to about 62 amperes, and the engine only indicated about 73 horse-power. The efficiency had thus increased to about 40 per cent.

The figures of the tests are as follows:

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Work.	Volts on dynamo.	Amperes.	Engine horse-power given out by dynamo.	Indicated horse-power of engine.	Efficiency.
Pumps running 24 revolutions and delivering 117 gallons per minute, 890 ft. high (= 31.5 horse-power.)	605	62	50.2	73.0	$\frac{31.5}{73} = 43.1\%$
Columns run off and suction clack-lids open, engine, dynamo, leads, motor and pumps (Friction).	410	27	14.8	28.6	
Engine, dynamo, leads and motor. (Friction).	No register	No register		18.4	
Circuit open, engine and dynamo. (Friction.)				11.7	
Belt of dynamo and engine only. (Friction.)				6.9	

The percentage of losses, as calculated from the indicator cards, are:

		Per cent.
Engine friction, ...	= 6.9 horse-power	= 9.4
Belt and dynamo friction, . . .	= 4.8 "	= 6.5
Leads and motor, ...	= 6.7 "	= 9.4
Motor, belt, gearing, and pumps empty, ...	= 10.2 "	= 14.0
Load of 117 gallons through 890 ft.,	= 31.5 "	= 43.1
Water friction in pumps and rising main, ...	= 12.9 "	= 17.6
	73.0 horse-power.	100.0

The engine doing above load indicated 73.0 horse-power. The friction of the water in the pumps and rising main is arrived at by subtracting the sum of the theoretical horse-power in the water delivered, and the total friction, from the total indicated horse-power of the engine; or, $73 - (31.5 + 28.6) = 12.9$. This is not quite correct. The friction diagrams are necessarily taken with no load on the pumps, and hence are all slightly lower than is actually the case when the load is on. From the known efficiency of the motor, the loss in the pumps and rising main has been found to be not less than 10 horse-power. The coal consumption of this plant was so much less than that which obtained with the compressed air-plant formerly used for the same work, that at my suggestion careful tests were made of the air-plant efficiency.

The conditions were singularly favorable for comparison. The superseded air-

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engine was still in position, and could be coupled to the same pumps. There were also two sets of compressors—a pair of horizontal compressors 20 inches by 4 feet, and a large vertical one. These were each put to drive the pumps, but since the pressure obtainable was only sufficient to raise the water about 530 feet, the column was tapped at the Stanley Main seam. Tests of the electrical plant delivering to the same place were also made, with the following results: The horizontal compressors gave a net result of 12 per cent., and the vertical one gave a little over 14 per cent., while the electrical system returned 44.4 per cent. The figures of the latter test are as follows:

Work.	Volts on dynamo.	Amperes.	Engine horse-power given out by dynamo.	Indicated horse-power of engine.	Efficiency.
Pumps running at 8 revs and delivering 39 gals. per minute to "Stanley Main," 530 feet = 63 horse-power.	171.0 173.4	47.5 47.5	10.9 11.0	14.2	$\frac{6.3}{14.2} = 44.4\%$
Pumps at normal speed with suction clack-lids open and column run off.	128 127	28.0 26.0	4.82 4.45	6.3	
Motor and first shaft only running approx. at normal speed.	111 105	20.0 20.0	2.96 2.80	4.8	
Dynamo running at normal speed with circuit open.				1.715	

These last tests were all made by the colliery engineer, Mr. E. Brown. The results obtained were so satisfactory that the pumps were at work only a fortnight before Messrs. Locke & Co. gave an order for a duplicate set of electrical plant, intending to use the new machines for both hauling and pumping. The pair of 20 inches by 4 feet compressors were removed, and foundations were laid in their place for the two dynamos, the old air-compressing engines, 20.5 inches by 4 feet, being arranged by suitable gearing to run them. At the same time a set of three throw rams, 6 inches bore by 12 inches stroke, were specially designed for dealing with the water. This new plant was nearly all finished and delivered at the colliery by the early part of the present year, but press of circumstances prevented the starting of the engines till late in July. The new dynamo, motor and pumps show a further decrease of power for the same work. The quantities average about as follows: Engine, 50 revolutions per minute; dynamo, 480 revolutions per minute; 690 volts and 59 amperes motor, 450 revolutions per minute; pumps, 34 revolutions per minute; water delivered, about 125 gallons per minute. The arrangement of the motor-house is very complete. A main shaft is run through the three houses containing the two pumps and the hauling gear. This shaft is fitted with clutches, so that either motor can be coupled to either of the pumps or the hauling-sheaves.

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Or both motors can be coupled to the shaft, and all or any of the machines attached. It is a most convenient and efficient arrangement, and prevents a trivial accident resulting in a serious stop to the mine.

The following are the particulars of the endless rope haulage: The main or belt rope makes three turns round the main driving sheave, and then passing round a tightening sheave, is led through a short tunnel to the main road. Here it gives motion to two horizontal sheaves, making about two and a half turns round each. These sheaves are coupled to two endless ropes by friction-cones, worked by screw-gear. The main rope is kept running at the desired speed, and the two working ropes are started and stopped as necessary. The main rope is 1 inch in diameter. The working ropes are both ¾ of an inch. They are all made of crucible steel. The sheaves are all about 4 feet 4 inches in diameter over the iron lagging. The coal now delivered is about 400 tons per day of eight to ten hours, the "Haighmoor" rope only being used. The motor is running at about half the arranged speed, and is doing less than half its designed output. The "Kirkthorpe" rope will be put on next week. Even with this additional load of some 300 tons per day, there is an ample margin for future increase of output. The arrangement could easily deliver 1000 to 1200 tons per day if necessary. No difficulty of any kind has been experienced with the hauling details. The ropes work with the regularity of clock-work, maintaining an almost constant speed whatever the load may be. The dynamo and motor are running most satisfactorily indeed; without looking at the ammeter, it is hardly possible to say whether the working rope is on or not.

Exception may be taken to the tests of these air plants on the score of the compressors being old and out of order. Indeed, I admit that the efficiency should be nearer 20 per cent. with plants of modern design in good order.

In comparison with compressed-air systems, the efficiency of an electric installation gives it an important advantage. I may refer, for instance, to the following test of the compressed-air system at the Chapin & Ludington mines, Iron Mountain, Mich. These mines are 3 miles from the falls which supply the power. There are four turbines at the falls, one of 1000 horse-power and three of 900 horse-power each; total, 3700 horse-power. The pressure is 60 pounds at 60° Fahr. Each turbine runs a pair of compressors. The pipe to the mines is 24 inches in diameter. The power is applied at the mines to Corliss engines, running pumps, hoists, etc., and direct to rock-drills.

The plant cost $100,000 (estimated). A test made February 20, 1888, gave 1430.27 horse-power at the compressors, and 390.17 horse-power as the sum of the horse-power of the engines at the mines. Therefore, only 27 per cent. of the power generated was recovered at the mines. This includes the loss due to leakage and the loss of energy in heat, but not the friction in the engines or compressors.

In other words, the efficiency of the system as it would be stated for

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an electrical plant, does not exceed 25 per cent. The figures above are a fair example of tests which are made at regular intervals.

I venture to think that an electrical plant, to give an effective return of 390 horse-power on the shafts, etc., at the mines, would not have cost more, in this case, than $60,000 at the outside.

I take the foregoing from a paper by Mr. R. P. Rothwell, in the *Engineering and Mining Journal* of January 5, 1889.

Prof. Kennedy has made a test of the Popp system of pneumatic transmission in Paris. The best efficiencies to be expected seem to be about 20 per cent., even with modern plants.

PECULIAR WORKING OF A BLAST-FURNACE.

BY N. B. WITTMAN, BIRDSBORO, PA.

(Washington Meeting, February, 1890.)

THE working of blast-furnaces is always of interest to many members of the Institute, for which reason I present an account of the working of the furnaces of the E. and G. Brooke Iron Company, of which I have had charge for the past four years. Although there are no phenomenal runs to record, there have been long periods of satisfactory work, and also periods—altogether too long—when one of our furnaces has behaved in a way which may be described as "nagging." All the conditions were apparently normal, but only a very light burden could be carried, which of course resulted in a high fuel-consumption to the ton of iron, as well as increased cost of labor.

The two active furnaces of this company have the following dimensions and equipment:

	No 2. Feet.	No 3. Feet.
Height,	55	60
Diameter at bosh,	15	16
" hearth,	8	7½
" stock-line,	10	9½
Height of bosh,	23	23
" tuyeres,	5½	5
Diameter of bell,	7	6
Front,	closed	open

Both furnaces are blown with I. P. Morris condensing-engines, sim

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ilar in all respects save one, the engine of No. 2 having a steam-cylinder of 45 inches diameter, while that of No. 3 is 50 inches. In both, the diameter of the air-cylinder is 90 inches, and the stroke 84 inches.

In both cases the stock is raised by an air-hoist.

In No. 2, the blast is heated by three Durham stoves of 28 pipes each, the pipes being 14 feet high and 8 inches inside diameter. No. 3 has two Kent ovens, 40 pipes in each, 14 feet high by 7 inches inside diameter.

The ore-mixture, which is practically the same for both furnaces, consists of 75 per cent. magnetites and 25 per cent. Lake Superior or foreign hematites.

About 40 per cent. of the magnetic ore is similar in character to that of Cornwall, except that it carries a variable amount of free calcite, ranging from nothing to 8 per cent. The sulphur in it is also very inconstant, the extreme range being from 1 to 4½ per cent., probably an average of about 2½ per cent. There appears to be no definite relation between the percentage of free calcite and sulphur, the latter being in the form of pyrites, agglomerated with the ore, while the former occurs in loose pieces, varying in size from grains of sand to masses weighing several pounds.

The general gangue of the ore is hornblende, with which the calcite is sometimes associated intimately, as it seldom is with the ore.

This ore is the only one used which is inherently troublesome to work.

The points of difficulty are, that, in smelting, the calcite being variable, it is impossible to flux it with any degree of certainty as to the composition of the cinder, and that, in roasting, a large portion of the sulphur is oxidized to sulphuric acid, and combines with the lime, forming calcic sulphate, so that it is only practicable to reduce the sulphur about one-half by roasting.

The ore disintegrates very rapidly, and there are several soft veins in the mine, so that a considerable portion of it is fine, like sand.

It is a well known fact that an ore carrying sulphur in material quantity cools the hearth of the furnace, probably by dissociation of calcic sulphate, and "buckshot" results, while the hearth is "built up."

The fuel used is anthracite, with as much coke (usually from 20 to 30 per cent. of the fuel-charge) as the price of coke will permit.

Dolomite of very uniform composition is used as a flux, for the reason that no calcite of good quality is available.

About 1.8 tons of ore are required to the ton of iron, and 0.8 ton

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of stone is added to form a cinder of approximately 35 SiO_2, 12 Al_2O_3, 33 CaO and CaS, and 20 MgO.

As an instance of satisfactory work may be cited the record of furnace No. 3 for nine consecutive months, during which time 18,422 tons of iron were made, being an average of 472.36 tons weekly, on a fuel-consumption of 1.19 tons to the ton of iron. The best week's output was 531 tons, made with 1.05 tons of fuel per ton of iron.

This furnace was filled and blown in as follows: Three stands of oak wood were put above the tuyeres, and the hearth was packed full of light dry pine, saturated with oil. On top of the wood was put, as a blank, 26 tons of coke, with about 2 tons of good gray furnace-cinder to flux the ash of the coke and wood. Ore-charges were then commenced with 2000 pounds fuel ($\frac{1}{2}$ anthracite and $\frac{1}{2}$ coke), 1300 pounds ore, and stone calculated to form a slag of 39 per cent. SiO_2. The burden was increased 200 pounds every six charges, until the stack was full, after which the fuel-charge was $66\frac{2}{3}$ per cent. anthracite and $33\frac{1}{3}$ per cent. coke. The furnace was lighted at the tapping-hole and tuyeres. The wood burned well, and as soon as bright coke appeared at the tuyeres the blast was put on, and everything went off smoothly. The volume of blast was increased so that in ten days the maximum was attained. This furnace continued in blast over three years; had all sorts of mishaps, due to break-downs in machinery, which necessitated long stoppages without preparation; yet always worked without trouble, and, for the greater part of the time, with very satisfactory economy in fuel. It will be remembered that the lines of this furnace are by no means modern.

Furnace No. 2 was blown in, as nearly as possible, in exactly the same way, and went off equally well; but the work for the first nine months of the blast—which is the present one—was vastly different.

The general features of the process were normal; the stock settled evenly all around; tuyeres bright; gas good and plentiful; cinder hot and of proper basicity; but there was no indication that a heavier burden could be carried. The charge at the time of blowing-in was: fuel, 2000 pounds; ore, 2600 pounds; stone, 1240 pounds. After a few days an attempt was made to add more burden, although the iron was only an ordinary gray-forge; but after the additional ore—60 pounds—came to work, the iron was closer in grain, and evidence was soon given that the furnace was overburdened. The only abnormal feature was low pressure of blast, the gauge registering only

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4½ pounds per square inch for two or three weeks after blowing-in, and gradually rising to 5½ pounds, where it remained for about three months. The usual pressure with ¾ anthracite and ¼ coke (the fuel-mixture then in use) is 7 to 8 pounds, and the low pressure indicated the possibility of working up the center, or "ring-scaffolding." To ascertain whether this was the case, and at the same time endeavor to correct it, a fuel-blank of 20 tons was charged, with scrap enough to keep the iron from becoming too gray.

When this had been in the furnace about ten hours, the speed of the engine was reduced three revolutions, so that there might be a tendency of the blast to work up the walls and loosen any matter which adhered to them, and get it into the hearth about the time the extra fuel reached there. The presence of the blank in the melting-zone was manifested by the cinder in about twenty hours after it was charged at the tunnel-head, and after 60 charges or rounds had been put in on top of it (the volume of which would occupy the space between the stock-line and bosh), showing that the melting-zone was in the region of the bosh, and that no considerable accumulations were on the walls. The working of the furnace was not at all improved by the blank, or changed in any way that was perceptible. During the time from blowing-in, November, 1888, to March, 1889, the best week's work was 445 tons of iron on 1.3 tons of fuel per ton of iron, and the average production for five consecutive weeks was 431 tons, requiring 1.3 tons of fuel per ton of iron. The pressure still continued low. After some time it was discovered that the collars connecting the pipes to the mains in the hot-blast stoves had "raised" in all the ovens, and that several of them were split, so that it became necessary to give them a thorough overhauling, which occupied the greater portion of three weeks. While these repairs were being made two ovens were used, and the volume of blast had to be diminished in order to keep up its temperature. It may be mentioned here, as a tale of misfortune, that when one oven was off, and about twelve hours before it could be made ready for service, a violent rain storm occurred; a water-conductor, which passed over one of the ovens, became disconnected at a joint, and the water poured on the roof of the oven and found its way between two plates, which had warped somewhat, in a sufficient quantity to crack a piece out of one of the pipes. The piece was large enough to cause the hot-blast valve to drop, and, the furnace being full of cinder at the time, the belly-pipes all filled. This necessitated a stop of about four hours to clean them out, after which we were obliged to run twelve hours

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with one oven of twenty-eight pipes and a blast-temperature of 600°. Finally, the ovens were all in good shape again, and some improvement was noted in the working over that immediately preceding the time when repairs were commenced, but not nearly so much as I had hoped for. The pressure had gradually risen to 7 pounds, but the fuel-consumption per ton of iron remained high, and it was not long until the production fell off to less than before repairs were made, and the fuel rose to still higher figures. The pressure remained at 7 to 7½ pounds per square inch.

During the months of July and August the results were most discouraging. The average production for five weeks during that time was 361.8 tons, each requiring 1.508 tons of fuel. The worst week's work was: iron, 356 tons; fuel-rate, 1.6 tons.

The only abnormal feature apparent was that, during casting, the cinder would run out very hot and fluid, separating perfectly from the iron, until about one-third of it had run; then a mass of "buckshot" would follow, often choking under the skimmer, and causing iron to run out into the cinder-runners. This was followed by about an equal amount of hot, fluid cinder entirely free from buckshot. At some casts no buckshot was made, but when it did appear it was almost invariably in the way stated. This was a very unusual thing. In our previous experience with buckshot it had made its appearance with cinder visibly cold, the hearth would soon build up, and iron would be thrown out of the cinder-notch during flushing. In this instance, however, the cinder which preceded and followed the buckshot was hot and fluid, the hearth did not build up, and no iron was thrown out of the cinder-notch. The appearance of the cinder was unusually good. It is our practice to ladle out a sample of each flush of cinder while it is running, and pour it into a scorifying-mould, so that buttons from the various flushes can be compared. At this time there was scarcely any difference noticeable between flushes. One sample would represent the entire day's work; yet at some casts two or three tons of buckshot would run out and at others none. The top of the furnace was as cool as could be expected with such a light burden—about 600°, when the fillers were at work, which would increase to 700° or 750° when they stopped for an hour.

No gas-analyses were made, but the gas was clean and good. No firing was at any time necessary under the boilers, and the bleeder was seldom opened. The slides on the gas-valves of the hot-blast chambers were seldom moved. Everything seemed right, yet the results showed that something was seriously wrong.

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The coke strike occurred about the time we were most troubled with buckshot, and it was necessary for several days to increase the proportion of anthracite to 87½ per cent. This resulted in less iron and more buckshot. I mention the fact, as it will have a bearing later on. During the first week in September a slight improvement was noticed; no change had been made in any particular, and there was no reason to think that it was more than a temporary relief, after which a relapse might at any time be expected. The relapse, however, did not come. There was no more buckshot, and the iron indicated that more burden could be carried. The iron, at the time when the improved working commenced, was peculiar. It was only a good gray-forge in grain, but there was an appearance about it difficult to describe, though familiar to those who grade iron by its fracture, which indicated that it was very much grayer than the grain would lead one to suppose. Burden was added in 40-pound lots, with stone in suitable quantity, and when it came to work, evidence was still given that a little more could be carried. This was done in 40-pound lots at first, and 80-pound after there seemed less danger of overdoing the matter, until 800 pounds more ore was carried by 2000 pounds of fuel than had been possible during the summer. The period of transition from light burden to heavy occupied about three weeks, during which time the iron only varied in grain from very gray-forge to close gray-forge. The volume of blast, measured by piston-displacement, was steadily maintained, and the production increased for the five weeks ending with October to an average of 464.6 tons of iron, made with 1.206 tons of fuel per ton of iron. The best week was 507 tons of iron, made on 1.18 tons fuel-rate. On October 1st, when the price of coke advanced, the proportion in the mixture was reduced to 20 per cent., with which the best work was done. It was further reduced to 12½ per cent., which had caused much trouble during the summer; but at this time it had no such effect, and 488 tons of iron were made in a week with 1.2 tons of fuel. After running two weeks, however, it was deemed advisable to return to 20 per cent. of coke, the production having dropped to 453 tons; and now the production averages 69 tons daily, and the fuel is fairly constant at 1.2 tons per ton of iron. During the summer, when the results were most distressing, the process appeared singularly uniform, and no evidence of scaffolding could be detected. Now there is no uniformity whatever; the stock slips sometimes from 4 to 6 feet at flushing; the cinder varies from bluish-white with stony fracture to black with vitreous fracture, yet is

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always hot and fluid. The iron varies from foundry to mottled between casts; and the cinder is but a poor guide, as the gray cinder sometimes makes close iron, and the dark glassy cinder does not always make hard iron. There is much more evidence of scaffolding now than there was during the summer, but the cast-sheet has a much healthier look.

It is likely that, apart from atmospheric conditions, the ordinary causes which affect irregularities of working are: arching of the stock, thereby forming a temporary scaffold; dropping of unprepared stock, which has been adhering to the walls as a fixed scaffold; and filtration of fine ore through the coarse stock, reaching the hearth in the state of ore and desiliconizing the iron, thereby lowering its grade.

I was unable to detect any form of scaffolding, and as the proportion of fine ore in the mixture has not changed, it is not likely that it influenced the process. When a furnace does not work economically, it is customary to say there is a scaffold; but it has seemed to me that this term is often applied to express an abnormal condition, the cause of which is obscure. Scaffolds, in order to exercise a material influence on the process, must occupy considerable space; and, although our means of estimating the working capacity of a furnace between the stock-line and the melting-zone are very crude, it is at least possible to ascertain whether there is a very considerable contraction in the cubic capacity contained between the stock-line and bosh or not.

In furnace No. 2, large fuel-blanks have been charged, from time to time, when it seemed desirable for any reason to do so; and after 58 to 64 ore-charges were on top of the fuel, its presence became apparent in the melting-zone. There has been no variation observed in this respect at any time during the blast. At one time, during a former blast, when the furnace was working badly, a fuel-blank would work after 27 to 30 charges were on top of it. It was not difficult to determine then that the melting-zone was high, and that a scaffold existed.

It has been my practice to maintain a uniform volume of blast, measured by piston-displacement, under all circumstances, and to distribute the current according to the manner in which the stock settles on top. If the settling is even, tuyeres of equal area are used all around. If there is a marked tendency to settle faster on one side than another for a few days, nozzles are put in the tuyeres under the low side, and the blast, flowing in the direction of least resist-

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ance, naturally goes in greater volume through the large tuyeres, and causes the furnace to work faster on the side which was high. In this way incipient scaffolds are moved, and dangerous ones are not likely to form suddenly.

The practice of reducing the speed of the engine whenever the cinder becomes sharp I have always regarded as likely to form scaffolds on account of alternately raising and lowering the melting-zone, and, as I could never see how heat could be generated by diminishing the rate of combustion, I have avoided it.

The working of the furnace in question has been a mystery to me, and this account of it has been written in the hope that it may meet the notice of some among the members of the Institute who have had similar experience, and, more fortunate than I, have ascertained the cause. It is not usual to proclaim one's poor work in all its details and confess ignorance of its cause; but if one cannot get light without opening the windows, they had better be opened, though other people may thus be enabled to look in.

Discussion.

Harry S. Fleming, General Superintendent Cameron Iron and Coal Co., Emporium, Pa. (Communication to the Secretary): I have been much interested in Mr. Wittman's paper regarding the peculiar working of No. 2 Furnace at Birdsboro, as we had a somewhat similar experience here. Our furnace has the following dimensions:

	Feet.
Height,	75
Diameter at bosh,	16
Diameter at hearth,	10
Diameter at stock-line,	13
Height of bosh,	27
Height of tuyeres,	6½
Diameter of bell,	9

At first we used a mixture of Centre county, Lake Superior, and New York State Clinton ores: but we are now using the Clinton ore with a very small proportion of Lake ore.

Shortly after blowing in, we were troubled with buckshot and a close iron, as well as high fuel, with other symptoms nearly similar to those at Birdsboro, and concluded that our fuel-charge was not large enough to give sufficient heat to the hearth. This was increased by one-half and then again by a similar amount, making just double the charge first used. Each time we had a noticeable change

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for the better and were able to increase the burden and blow harder. The product also showed a marked improvement in both quantity and quality.

Only once since that time have we been troubled with buckshot and the peculiar forge-iron described by Mr. Wittman: that was when working a very irregular mixture, with a cinder containing 39 per cent. of silica, and very dirty coke. The iron made from this had a hard, brittle fracture and pitted face, but contained 3.26 per cent. of silicon. The burden at this time was 1 fuel to 1.46 ore yielding 45 per cent. Burden and limestone were increased, and the conditions became normal. We are now working a burden of 1 fuel to 1.87 ore yielding 41 per cent. of iron in the furnace, using local coke.

I am not familiar enough with the working of anthracite furnaces to say what was the cause of Mr. Wittman's trouble; but, judging from my experience, I would say that an increase of limestone and, if necessary, of the fuel-charge, would have produced beneficial results.

Many furnacemen seem afraid to use limestone in sufficient quantity, and no doubt it is somewhat risky with a very variable mixture, and especially in anthracite practice; but here we have run for several weeks, making an open-grained, dark and strong foundry iron with an excellent reputation on the market, while our cinder would slack on the yard and contained as low as 30 per cent. of silica.

MR. WITTMANN (Communication to the Secretary): In reply to Mr. Fleming's statement concerning the beneficial results of increasing the fuel-charge in the blast-furnace, I would say, that it has been my practice for some time to charge 5000 pounds of fuel, and put on top of it 8 barrows of ore and 4 of stone. Our practice for a long time, was to charge only 4000 fuel, but we noticed some improvement after the larger fuel-charge was tried, and therefore continued it. This had no bearing whatever, however, on the work recorded of No. 2 furnace, for the reason that the charging was the same during the period of poor work as it has been since the good work commenced. The cinder was as basic at the time we were troubled with buckshot as I dared allow it to be. When a furnace is making buckshot and the cinder is unduly basic, the slightest cooling of the hearth is apt to result in losing the iron-notch. My practice is to carry a very basic cinder when the hearth is going down, and a neutral or acid cinder when it is building up. The peculiar feature of the buckshot period in the case under discussion was, that the hearth showed no tendency to "raise." Several gentlemen have

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written me their opinions on the causes which produced the change in the working of the furnace.

A thinks the furnace was scaffolded, and when the scaffold worked off the improved working commenced.

B thinks there was insufficient blast.

C says the bell and stock-line are too large in proportion to the bosh, and suggests a 6-foot bell.

D thinks that there were dirty walls, or dirt-scaffolds—not necessarily occupying much space, and therefore not to be detected by blanking.

Of all the suggestions made, I should incline most to the theory of dirty walls, except for the fact that the conditions seemed so uniform at the time—the cinder being remarkably constant in heat and basicity, and the rate of driving almost clock-like—that it is difficult to believe a scaffold of any kind was forming or being moved.

The conditions which determine the ore-reducing and metal-carbonizing power of a unit of fuel are somewhat obecure.

NOTES ON AMERICAN CANNEL COAL.

BY GRAHAM MACFARLANE, C.E., LOUISVILLE, KY.

(Washington Meeting, February, 1890.)

THE largest area of high-grade cannel coals known in America is doubtless found in eastern Kentucky. Notwithstanding the popular belief to the contrary, there is no reason to doubt that these eastern Kentucky cannel-fields, following the general rule, will be uncertain and treacherous. But, taken in the aggregate, their extent is very great.

The principal service to which cannel coal can be put is enriching illuminating-gas. The electric light has, to a certain extent, created a demand for better gas, and has increased the market for gas-enrichers. Before the discovery of rock-oil, coal-oil was distilled from cannel coal and cannel shale. This business was getting fairly started when the cheaper rock-oil, or petroleum, overcame it, and the great distilleries went into disuse and ruin. Now, again, the chief competitor with cannel coal is oil, namely, the cheap oil of Lima, Ohio. Under proper treatment the vile odor of this oil can be removed.

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Recently American cannel has been successfully marketed in Italy and South America; and doubtless, with the natural extension of our commerce, new markets will be developed for really first-class American cannels. My belief is, that West Virginia and Kentucky will supply these markets. There is a large production of inferior cannel in Ohio, Indiana, and Missouri, which is, and will continue to be, sold in the adjacent markets.

The following is a list of cannel coals now produced for tidewater and export markets:

I. The "Breckinridge," mined in Hancock county, western Kentucky:

Analysis by State Chemist.

	Per cent.
Volatile matter,	60.90
Coke,	39.10
	100.00
Sulphur,	1.890
Ash,	12.100
Thickness of seam,	0 to 30 inches.
Shippers, Breckinridge Co., Limited, Louisville, Ky.	
Annual product,	15,000 tons.

II. The "Grayson" or "Old Kentucky Boghead," mined in Carter county, Kentucky:

Analysis by State Chemist.

	Per cent.
Volatile matter,	66.30
Coke,	33.70
	100.00
Sulphur,	1.32
Ash,	4.80
Thickness of seam,	15 to 20 inches, cannel. 20 to 24 inches, bituminous.
Shippers, Macfarlane & Mordue, Louisville, Ky.	
Annual product,	12,000 tons.

III. The "Chattaroi," mined in Johnson county, eastern Kentucky:

Thickness of seam,	16 to 36 inches, cannel. 6 to 12 inches, bituminous.
Shippers, Macfarlane & Mordue, Louisville, Ky.	
Annual product,	12,000 tons.

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IV. The "Whitehouse," mined in Johnson county, Kentucky. This is the same cannel as the "Chattaroi," produced on the adjoining property:

Analysis by State Chemist.

	Per cent.
Volatile matter,	40.20
Coke,	59.80
	100.00
Sulphur,	0.956
Ash,	8.806

Shippers, Whitehouse Cannel Coal Co., Ashland, Ky.
Annual product, 12,000 tons.

V. The "Cannelton," mined in Kanawha county, West Virginia:

Analysis.

	Per cent.
Volatile matter,	43.10
Coke,	56.90
	100.00
Sulphur,	1.162
Ash,	7.40
Thickness,	0 to 36 inches.

Shippers, Cannelton Coal Co., Cannelton, W. Va.
Annual product, about 20,000 tons.

VI. The "Jellico," mined in Campbell county, Tennessee:

Analysis.

	Per cent.
Volatile matter,	49.85
Coke,	50.15
	100.00
Sulphur,	.748
Ash,	15.12
Thickness,	0 to 36 inches.

Shippers, Standard Coal and Coke Co., Newcomb, Tenn.
Annual product, about 12,000 tons.

Several railroads are now being built into the coal-fields of eastern Kentucky, and much money will be invested in coal-lands. Those investments which are based on careful examinations of the coal-fields by competent engineers and equally careful examinations of the titles by competent lawyers, will doubtless prove very profitable. The titles are in such a condition that the greatest care is requisite; and, generally speaking, such titles only should be accepted as are accompanied by *actual possession* of the land.

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A PROPOSED METHOD FOR WORKING TELLURIDES.

BY FRANK CLEMES SMITH, ANN ARBOR, MICH.

(Ottawa Meeting, October, 1889.)

THE occurrence of the tellurides of gold and silver, even in small quantities, is so rare that their metallurgical treatment has engaged little attention. A residence of several months at one of the few mines in North America the product of which consists entirely of such ore, has led me to arrange a method for its economical treatment. The mine referred to is the Huronian, situated a few miles distant from Jackfish Lake, in Ontario, Canada. The plan proposed is for this peculiar case, and is put on record as possibly suggestive and helpful in any similar instance, should it occur. Since freight-rates from the mine to the nearest railroad-point were extremely high, and for other economical reasons, the method proposed is modified by the use, as far as possible, of the machinery already at the mine, and is so arranged that bullion instead of concentrates would be obtained as a final product.

The ore-vein at the Huronian mine varied in width from about 4 feet at the surface to 2 feet and less at the lowest level—125 feet—and dipped southeast about 88°. The rock consisted, on both foot- and hanging-walls, of soft, green Huronian slates carrying much highly crystalline but barren pyrite. The ore carried an inconsiderable amount of galena with iron and copper pyrites, argentite, hessite, sylvanite (or petzite), and probably altaite, the telluride of lead. Near the surface the ore showed considerable native gold, but this disappeared with greater depth. Two complete analyses of minerals containing tellurium are here attached, as they establish the presence of two important compounds:

I. Analysis of a mineral supposed at the mine to be sylvanite, but agreeing more closely in composition with petzite $(Ag, Au)_2 Te$:

Ag	= 41.062
Au	= 23.691
Te	= 32.007
Pb	= 0.071
SiO_2	= 0.097
Fe	= trace
	96.928

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The specific gravity of this mineral was 8.103 and the color was steel-gray.

II. Analysis of a telluride of silver:

Ag	= 62.01
Te	= 29.62
Fe	= 1.76
Au	= trace
	93.39

The sample was probably hessite (Ag_2 Te), in which some of the tellurium was probably lost in the analysis.

In some cases the tellurides existed in irregular, bluish-gray masses enclosed in the pure quartz; at other times they occurred as very dark brown scales, filling small seams in the quartz. Samples of quartz from this mine, which showed no trace of metallic contents, but seemed to be saturated with the tellurides and resembled an ordinary piece of smoky quartz, have given upon assay as high as 902 ounces of silver and 34 ounces of gold per ton. Through the kindness of the manager of the Huronian mine, Mr. Thomas A. Keefer, of Port Arthur, Ontario, I am permitted to publish the figures obtained in an experimental run of one week upon pure vein-rock from the mine. The values given to the various products are probably all too high, owing to the lack of a proper assay-balance, but, as they are approximate and not *too low*, they will suffice for the purpose.

The ore was crushed, wet, in two five-stamp batteries, amalgamated on copper plates, and concentrated upon three Frue vanners, each vanner being provided with an amalgamated copper plate on its distributor. The ore crushed in 5½ days, including night run, amounted to 46.7 tons, or 93,400 pounds, its assay value per ton being $57.13, or its total value $2667.97.

YIELD.

Vanner No. 1.	Concentrates,	3892 lbs.,	@ $284.45	per ton,	$553.59
" "	Slimes,	70 lbs.,	@ 600.00	"	21.00
" No. 2.	Concentrates,	1276 lbs.,	@ 224.00	"	142.91
" "	Slimes,	40 lbs.,	@ 553.00	"	11.06
" No. 3.	Concentrates,	650 lbs.,	@ 147.00	"	47.77
" "	Slimes,	25 lbs.,	@ 414.00	"	5.17
Total,		5953 lbs., or 2.976 tons, worth,			$781.50
Gold taken from the plates,					41.20
Total value,					$822.70

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Loss, 69.1 per cent. of ore-value.
Average value of concentrates per ton, $262.60.
Yield per ton of crude ore treated, $17.61.
Concentrates represent, *in bulk*, 6.35 per cent. of the crude ore treated; *in value*, 29.3 per cent.
Work per day (24 hours) of ten stamps, 8.49 tons.

An explanation of the extraordinary loss in the above test-run is not difficult. The machinery used was purchased at second-hand from an abandoned enterprise near Port Arthur, with the hope, on the part of the management, that it would suffice to treat the ores of the Huronian mine, the peculiar character of which was not understood. The ore treated came from some distance below the surface, and carried but little free gold, as is shown by the value of the amalgam saved from the coppers—something less than $1 per ton of ore treated. The tellurides of gold and silver are extremely sectile, and this property alone is sufficient to render futile the method used, since much of the valuable part of the ore could not be saved by the Frue vanners, either in the concentrates or in the slimes, but floated off with the water and tailings as an impalpable powder. This is proved by the comparatively high value of the slimes, or that part of the pulverized ore which was not washed off the belt, but which was still light enough, when disengaged from the heavier ore-particles, to float upon the surface of the water, and thus pass into the slime-boxes.*

It is evident that the method employed for treating the ores in question was decidedly the wrong one in view of the yield obtained. But even if the concentrates could have been made to contain the full assay value of the ore, deducting the gold found on the coppers, the difficulty of transportation to shipping-point made it a mistake to aim at producing anything but bullion.

Although the precious-metal contents of these ores might be saved by a smelting with lead, the cost of obtaining enough lead to supplement the very small percentage in the ore, the uncertainty whether the extent of the ore-deposit, when thoroughly explored, would warrant so extensive and costly a plant, and the uncertainty as to the

* Although it is stated that the Frue vanner was invented for the purpose of concentrating tellurides, and the machine is advertised by the makers to be capable of performing that duty, the above experience would seem to contradict its claims to efficiency. In the test mentioned, the vanners were most carefully set up, constantly examined and adjusted by a man who had worked with them for many years, and who was recommended by the makers themselves.

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existence of any neighboring deposits of smelting-ores, rendered the method inapplicable in this case.

The processes used in the extraction, in the wet way, of the precious metals from their sulphuretted ores generally make use of a primary roasting, either with or without fluxes. Since the telluride of gold is volatile at and below a roasting-heat, the most carefully arranged dust-chambers would scarcely avoid a very large loss. It is probable that no process of amalgamation could be applied with good results to these ores, at least when *raw*, since mercury attacks the gold of tellurides only after prolonged digestion, and then only to a limited degree. It may be said that tellurides do not amalgamate.

We find ourselves subject to loss of precious metals in two principal operations, viz., concentrating and roasting, and, at the same time, are limited in our choice of a method to some wet process. As the ore is of too low a grade to treat as a whole, it *must* be concentrated. Therefore, a plan is suggested to modify the mode of concentration used so as to reduce the loss to a minimum.

Thus far the saving of the gold and silver only has been considered, but there is still a valuable metal in these ores, *i.e.*, the tellurium, of which Mr. T. A. Edison says, in a private communication to the writer: "I have worked it to a considerable extent, and I pronounce it a marvellous substance; I could use it in large quantities if it were cheaply produced." Its great cost and its scarcity have, doubtless, retarded the knowledge of its valuable properties. About forty years ago Wöhler wrote largely and enthusiastically upon its organic compounds and its pharmaceutical uses. It possesses properties similar to those of selenium, which changes its electrical resistance under the influence of light. It may then be advisable to attempt to save the tellurium.

As already stated, no attempt is made to decide upon the *best* method for treating the ores in general which carry tellurium, or even to decide upon the best method, in the abstract, for treating ores peculiar to this mine; but rather, for various economical reasons, to make use, in part at least, of the apparatus already at the mine by supplying necessary modifications, and finally to produce fine bullion.

Various methods of treatment of the tellurides of gold and silver have been suggested and actually used, a few of which are referred to. With the exception of the last two, however, they contain evi-

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dent faults, or would for other reasons be unfitted for use in this case.

(1.) In Transylvania, the tellurium has been recovered by fusion of the pulverized ores with sodium carbonate and subsequent separation from the soluble sodium telluride. The other metals were afterwards extracted from the residue.

(2.) Boulder County, Colo., has produced large quantities of tellurides. In the neighborhood of Gilman such ores are found in flat veins in the quartzite. These ores are either mixed with others and smelted with lead or are sent to the Boston and Colorado smelter at Argo where everything goes into a copper-matte.

(3.) Some years ago a method for the treatment of tellurides was recommended by Mr. C. A. Stetefeldt.* The ore was to be crushed and passed through rolls, then screened and three products obtained of which the first or coarsest was to be concentrated in jigs, the second or finer product was to be concentrated on joggling tables and the third product or fines was to be concentrated upon rotary buddles. Of the three products of concentration the richest was to be cupelled with lead—the ore being added gradually. The tellurium, according to Mr. Stetefeldt, would partially volatilize and in part become PbTe and form "abstrich" with the PbO. When this abstrich ceased to form, the auriferous and argentiferous lead was to be drawn off and cupelled upon another test. The abstrich was to be melted in a reverberatory furnace, to remove PbO, and the residue from this process worked over, to save the tellurium.

In Mr. Stetefeldt's process the proposed concentration of the fines upon round buddles would undoubtedly prove a fatal defect.

(4.) The Walker-Carter vapor-amalgamation process is said, by Mr. E. N. Riotte, in a private communication to the writer, to be capable of saving the gold and silver from tellurides.

(5 and 6). Finally, the Hauch process † has been successfully used, as has also the process described by Schröder,‡ the latter with the tellurides of Nagyag, Transylvania; and an adaptation of these methods is proposed for the treatment of the ores of the Huronian mine.

The accompanying drawing gives an arrangement of the apparatus proposed for the separation of the fine tellurides and the concentration of the coarse remainder. The dry ore is broken in the crusher,

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* *Berg-und Huettenmaennische Zeitung*, 1865, vol. xxiv., p. 374.

† *Oester. Zeit.*, 1876, No. 23. ‡ *Oester. Zeit.*, 1872, No. 28.

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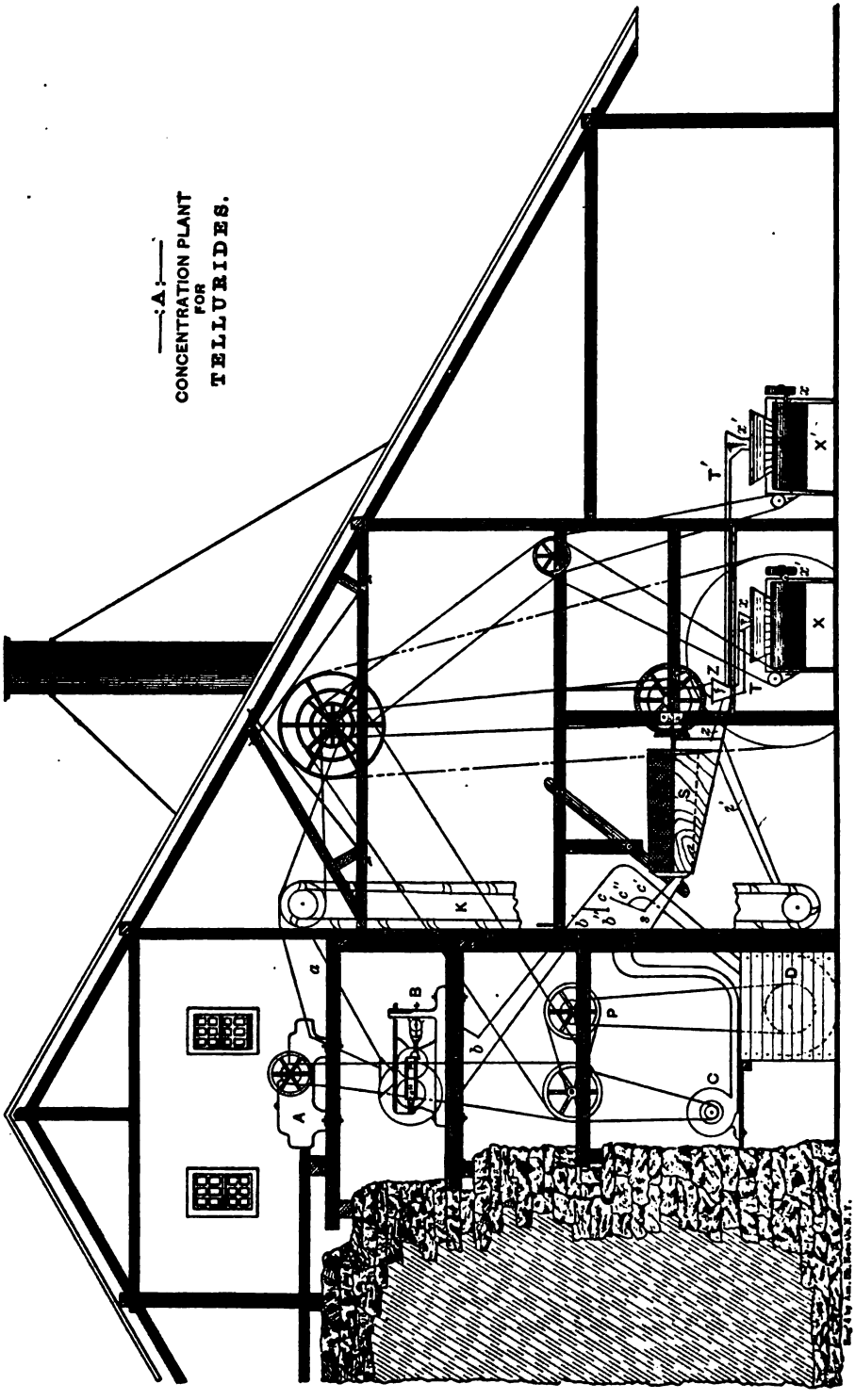
—:A:—
CONCENTRATION PLANT
FOR
TELLURIDES.



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A, and passes thence, through the chute, *a* (which is carefully closed in to prevent loss of the fines), to Cornish rolls, *B*. From the latter it slides in the covered chute, *b*, but, on falling vertically from the point, *b'*, the stream of ore meets, at the point *b''*, a blast of air from the blower, *C*. The dust from the pulverized ore is blown through the opening *c*, and falls into the chute, *c'*. It would seem advisable that the air-blast be so regulated as to deflect and throw against the inclined screen, *c''*, all of the fines which are yet too heavy to be blown over it through *c*. The screen, *c''*, may be made of 40- to 50-mesh wire-gauze, and the portion of fines passing through it joins the dust in chute, *c'*, and passes with it into the tank, *D*. This tank may be lined with lead and filled with a solution of dilute HCl to such a level that the nose of the chute, *c'*, dips just beneath it. This arrangement causes the blast of air to perform the additional service of partially agitating the solution of HCl. The latter, being an absorbent medium for the fines, has the further function of preparing this part of the ore for the ensuing operations. The tank, *D*, should be provided with a cylindrical bottom in which any convenient form of stirring apparatus may be caused to revolve by the pulley, *P*. It should be provided also with a proper outlet for the withdrawal of the pulp.

The portion of the ore which has resisted the deflecting action of the air-blast continues on its way and passes through the chute, *s*, into the revolving wire-cloth screen, *S*. Its mesh must be determined experimentally.

The ore passing through it goes through the chute, *z*, into the sizer, *Z*. The coarser portion which does not pass through the screen is conducted by the chute, *z'*, to the conveyer, *K*, which returns it to the rolls, *B*.

Water is admitted to the sizing apparatus through the cock, *W*, and the two sizes of pulverized ore pass through the troughs, *T* and *T'*, respectively, and are deposited upon the distributors, *x* and *x'*, of the Frue vanners, *X* and *X'*.

By the above arrangement only the ore-particles large enough to escape the blast of air, or of a size greater than 40- to 50-mesh, reach the vanners. They are all too large and heavy to lead to serious loss in concentration.

The particular point at which the blast of air is introduced, its pressure, and the choice of the most convenient size and weight of screens, are all matters to be regulated by trial. The concentration

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should be so conducted that the concentrates and fines together should represent not over 10 per cent. of the ore treated.

From a knowledge of the character of the ores it is believed that, after being crushed dry as in this case, the greater portion of the sectile tellurides will be collected as dust in the bin, *D*. If this be the case we may treat the concentrates from the vanners by one of the following methods:

1st. Roast, if the remaining amount of tellurides is not found to be too great; pass through the stamps and amalgamate, or treat by such method as the character of the valuable mineral-content renders advisable.

2d. Roasted or raw, as may be found best on trial, the concentrates may be finely pulverized and treated with the dust from bin, *D*.

The fines, after withdrawal from bin, *D*, in the form of pulp, may be agitated and mixed together in a heated solution of dilute HCl, this drawn off and the pulp further agitated with hot, concentrated HCl until no more H_2S escapes, the operation being performed in a lead-lined vessel or tank.

Chlorine may be generated directly in this solution by use of oxide of manganese, or the solution of concentrated HCl may be drawn off for use with the next charge of fines, and the pulp treated with aqua-regia. The tellurides are dissolved with the other minerals, the solution is drawn off and the residue washed. The lead and lime may be removed from this solution by a small addition of H_2SO_4. The solution is then decanted from the precipitate and the gold thrown down by means of $FeSO_4$. After filtration the tellurium may be precipitated with metallic zinc and purified by treatment with chlorine and H_2SO_4.

The chloride of silver left in the pulp-residue may be decomposed by means of zinc and HCl, the metallic silver dissolved by H_2SO_4 and re-precipitated from the sulphate solution by means of metallic copper.

The process above outlined seems to do away with the loss in concentration and also with a part of the loss in roasting. If much of the tellurides should remain in the concentrated portion of the ore, the roasting ought to be avoided if possible; for a series of small muffle tests upon a rich sample of telluride-bearing quartz satisfactorily proved the inevitability of loss by volatilization. The experiments were conducted at various heats, with and without salt, and in every roasting loss occurred, commencing even before the sample became red. Where the loss by volatilization is so very

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great it seems dangerous and expensive to trust entirely to condensing-chambers. No experiments were made to show the loss of gold in roasting the tellurides with admission of superheated steam. In laboratory-experiments with the same ore, the gold, silver and tellurium were separated without difficulty by means of the method described.

THE ASSOCIATION OF GOLD WITH OTHER METALS IN THE WEST.

BY RICHARD PEARCE, ARGO, COLORADO.

(Presidential Address at the Washington Meeting, February, 1890.)

In looking around me for some subject on which to frame an address for this meeting, it has occurred to me that some results of observation and investigation, in regard to the varied conditions under which gold is found, may possibly prove of interest.

The subject is one which frequently crops up in discussions at our meetings, but there is always an immense amount of uncertainty expressed as to the actual condition in which the gold exists in any particular ore.

It follows, therefore, that any light which might be shed on this rather complex subject, would be of value to science and of service to mining and metallurgical practice.

In some investigations which I have, from time to time, been led to make, for the purpose of determining to what extent an ore might be concentrated, I have been astonished at the very varied and apparently inexplicable results encountered, and have been naturally led to inquire into the causes for such seeming inconsistency.

The mines of Gilpin county, Colorado, where gold was first discovered in that State, furnish an infinite variety of examples of its occurrence, and it was here that difficulties first presented themselves to the Colorado miners, in the sudden transition from what is called "free-milling gold-ore" to an ore characterized by properties which make the ordinary treatment by amalgamation in stamp-mills utterly impracticable. So long as mining was limited to what might be called the zone of oxidation in the veins, just so long might amalgamation be conducted without difficulty; but directly the miner reached a point beyond the influence of the oxidizing power of the surface-waters, the gold resisted all attempts at amalgamation, and recourse was had to smelting.

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The question naturally arises, how such a mineralogical change as has been described, affects the treatment of the ore by amalgamation. The ores above the water-line may be considered as made up of silica, peroxide of iron and an aluminous material formed, in all probability, by the decay of the feldspar in the adjoining rock, which rock is altogether gneiss. The matrix of the gold in these zones is, in fact, a veritable gossan, the ferric oxide of which has been formed by the oxidation of the pyrites which, at one time, constituted a large percentage of the total contents of the vein.

This change from pyrite to limonite must have produced some sort of cleansing effect, so far as the gold was concerned, and it must have left this metal in such a form as to be readily attacked by quicksilver.

It is, I think, not improbable that the gold, during the change, was dissolved and again precipitated in a pure state, or in a form, at any rate, free from those impurities which may have existed in the original association with pyrite and other sulphides.

In a paper read before the Colorado Scientific Society, Mr. R. G. Hills distinctly points to an example of this kind, which he has seen in the Little Annie mine, in the San Juan district, and he gives evidence indicating that gold has been dissolved, or leached out from one point, and re-precipitated at another, as a result or concomitant of the alteration of pyrite into limonite. In my opinion, such a change can be explained without much difficulty, and without even taxing one's belief, to any great extent, in the solvent action of certain solutions, said to attack gold. The presence of alkaline chlorides (a natural constituent of most mineral waters) and manganese peroxide (which always occurs in gossans) will be sufficient with the aid of free sulphuric acid (which is always formed by the oxidation of pyrite) to bring about such a change as has been referred to.

This has been conclusively proved by some experiments which I made a few years ago.

If we examine carefully into the gold-veins of Gilpin county we find that they are composed, below the water-line, of quartz, pyrite, chalcopyrite, sphalerite and galena, with an occasional sprinkling of fahlerz and chalybite.

In all my experience, which, I may say, has been obtained by an almost daily examination of these ores for eighteen years, I have never seen but one small characteristic specimen of arseno pyrite. Nevertheless, in roasting Gilpin county ores, an appreciable amount

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of arsenic is always found; so that arsenic may be said to be present in these ores in some form which does not make itself manifest as belonging to any mineralogical species, but rather as an accessory impurity.

A microscopical examination of the ores, after careful concentration by washing, shows the gold to exist in fine particles. Its color, however, is usually paler than that of the gold taken from the oxidized portion of the vein.

It has been generally supposed that the auriferous pyrites of Gilpin county contained no tellurium (which, if present, might account for any mineralizing action on the gold), but since my discovery of tellurium in the silver-bearing pyrites of Leadville I have felt that tellurium might perhaps be found in sufficient quantity to account in some measure for the "baseness" of the Gilpin county ores.

Bismuth, I have, for many years, known to be present in these ores; and I have attributed many peculiarly troublesome features in their metallurgical treatment to what may be an intimate association with bismuth and arsenic.

A recent careful examination of a piece of rich gold-ore from the Gregory mine, brought to light the existence of both tellurium and bismuth, in well-marked quantities; and although it may be difficult to prove that bismuth is combined with a portion of the gold in the ore, I, personally, have not the slightest doubt that such is the case.

My conclusions are based on the fact that a definite, crystallized compound of gold and bismuth may be prepared, artificially, as was shown by some experiments of mine, embodied in a paper read before the Institute, and published in the *Transactions*, xiii., 738. Moreover, within the last few months, a notice has appeared in the *Engineering and Mining Journal*, announcing the discovery in Australia of a mineral consisting of a crystalline compound of gold and bismuth, having nearly the same atomic composition as that of the compound which I personally prepared.*

With such facts as the above before us, I think it not unreason-

* Since this paper was written, my attention has been called to notices in the *Am. Jour. of Science*, in 1847 and 1857 (2d series, vols. iv. and xxiv., particularly iv., 281), and Shephard's *Mineralogy*, 1857, p. 304, where native alloys of bismuth and gold are described. Prof. Shephard calls the mineral bismuthaurite. Dana's *Mineralogy* (1882), p. 199, barely mentions, as maldonite, gold containing traces of bismuth. See, also, in the paper presented by Mr. S. F. Emmons, at this meeting, on the Gold Deposits of Montgomery County, Md., mention of the occurrence of bismuth-compounds in the South (page 391 of this volume).

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able to assume that a portion of the gold contained in what is termed refractory ore, exists in combination with one or more of these metals, and in a form which resists the solvent action of quicksilver.

A great deal of uncertainty exists as to the precise part which arsenic plays by its association with gold in the ores; but, since the discovery in Canada of sperrylite (arsenide of platinum), I have felt that possibly a similar compound of arsenic with gold might exist.

In order to test the effect of arsenic vapor on gold, I have heated thin sheet-gold to a red heat, and then passed the vapor of arsenic slowly over it, in an atmosphere of hydrogen. The surface of the gold showed a distinct change of color; a very fine grayish coating being apparent, which may have consisted of a compound of gold with arsenic.

A microscopical examination of auriferous arseno pyrite from California showed the existence of metallic gold; yet, although these evidences were abundant, there was scarcely a particle which showed the least brightness or luster; their surfaces appearing of a dull grayish-black color, and, in some instances, having a distinct bronze appearance.

In a discussion following the paper by Mr. Blow, on the character of the Leadville ore-deposits, read at the Colorado meeting of this Institute last June, Mr. Blow declared that gold occurs in certain of the Leadville ores in some other condition than that of free gold. He, however, had not been able to ascertain the precise nature of the peculiarity. Since the time of that discussion, I have accidentally come across some interesting facts which may, perhaps, throw some light on the subject.

The Boston and Colorado Smelting Company has been, for some time past, a large purchaser of Leadville pyrites containing silver and a small value in gold. The Louisville mine produced a large quantity of almost pure pyrite, having a coarse-grained crystalline structure; and it puzzled me not a little to find a pyrite of such character containing any precious metal of value. It has been the practice, for some months past, to roast this ore in kilns; and on the surface of these, a few months ago, I noticed, upon some pieces of pyrite which had remained unaltered (owing to the temperature at that point being too low), brilliant crystals of what proved to be nothing more or less than metallic tellurium.

Here we have demonstrated, beyond a doubt, the existence of tellurium in an ore in which there was not the slightest suspicion of its occurrence. Is it not reasonable to suppose that this metal acts

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as a mineralizing agent, not only in relation to the small amount of gold which the ore contains, but also in its combination with silver as telluride of silver?

The mineral hessite has been found in large quantities in one or two of the mines at Red Cliff, a mining district situated some distance northwest of Leadville; but the wholesale distribution of tellurium in masses of pyrite is certainly a new feature in the composition of what are called Leadville sulphides.

The association of gold with bismuth and tellurium in ores said to be of a refractory nature, is encountered also in the mines of Shasta county, California. A sample of ore from a mine at Old Diggings, near the town of Shasta, was found to consist of a mass of quartz having small cavities filled with a vitreous variety of iron oxide, which had undoubtedly been formed by the alteration of pyrite. A concentration of this ferruginous material by washing showed the gold to be intimately associated with it. The ore in bulk assayed 17 ounces per ton, while the concentrated product assayed 109 ounces per ton.

An examination of this material showed the existence of very well-marked quantities of tellurium and bismuth. Under the microscope the gold exhibited that rusty coating which has been described by Dr. Egleston in his paper on "The Cause of Rustiness, and of Some of the Losses in Working Gold" (*Trans.*, ix., 646), and probably the refractory character of this ore may be partly due to this cause. I have reason to believe, however, that the presence of tellurium and bismuth has some influence in the treatment of such ores. It is quite a common thing in the West for the miners to attribute many of their amalgamation troubles to "rustiness," when, if these troubles were better understood, they might be explained by the fact that other agents were present, extremely minute in quantity, perhaps, and most subtle and insidious in behavior.

I may cite other examples of characteristic Gilpin county ores which I have examined.

A piece of ore was recently broken at a depth of 1900 feet in the California mine, the deepest point yet reached in the whole of that district. It assayed 3 ounces of gold and 30 ounces of silver per ton, and was composed chiefly, so far as the eye could discover, of an admixture of quartz, pyrite, chalcopyrite, fahlerz, zincblende and chalybite. After concentrating to a very high point by washing, the concentrates showed no traces of gold whatever, even under the microscope.

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A chemical examination of this specimen gave the strongest evidences of bismuth. I have it on the authority of the manager of the mine that he has not been able to discover any free gold by the usual operation of panning. I find, however, that after prolonged boiling with nitric acid, gold is separated in fine crystalline particles, which may have existed, combined with silver, in a matrix of bismuth, under conditions somewhat analogous to the gold and silver crystalline alloy which I have produced artificially.

A sample of ore from the Main mine, in Gilpin county, which was of the ordinary pyritous character, exhibited strong bismuth reactions, and a specimen of what is called "surface-ore," from the Tierney mine, in the same district, in which the pyrites had been oxidized to a large extent, proved to contain both tellurium and bismuth.

I may observe, that in all the examinations I have made on Gilpin county ores bismuth was found in every instance, while tellurium was discovered in two cases. It must be understood that the bismuth in these several experiments is intimately associated with the gold, as in concentrating by vanning only the heaviest material, having a specific gravity above galena, was retained for investigation.

One other example, referring to an examination of gold in some rich specimens from the Ontario mine near Breckenridge, Colorado, is of special interest on account of its freedom from pyritous ores of any kind. On the other hand, it exhibits marked indications of what is termed rustiness.

The Ontario mine produced, some years ago, beautiful crystals of native gold. These were found in small seams, at the junction of porphyry and slate, and were embedded in a matrix consisting of hydrated oxides of iron and manganese. An examination of this ochreous material showed the existence of both bismuth and tellurium; the former in well-marked quantities. On boiling for some time with nitric acid and sulphuric acid a beautiful purple color was produced, having all the characteristics of "purple of Cassius." Here we have evidence of the existence of gold in a very finely divided condition, and in all probability combined, before being treated with acid, with bismuth or tellurium, or both.

I have observed this color in two or three cases when investigating material very rich in gold; and it seems to correspond with the results obtained from the examination of artificial alloys of bismuth, gold, and copper, except that, in the latter instance, the "purple of Cassius" was present in greater quantity.

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I may have attached too much importance to the association of these two metals, bismuth and tellurium, with gold. We know, however, that tellurium can by no means be regarded as an accidental associate of gold, and I feel thoroughly convinced that if this investigation, which I have begun and in which I have done so little, be followed up, it will be proved beyond a doubt that the metal, bismuth, performs a by no means small part in influencing the general disposition of gold, and also in causing the refractory behavior of this metal, under metallurgical treatment.

The occurrence of gold seems to be intimately associated with that of eruptive porphyries, and the impregnation of veins with gold-bearing minerals is, apparently, always accompanied with silica. Evidence of intense thermal action is one of the chief characteristics of gold-deposits. Examples of these processes may be found in Gilpin county, and also in Boulder county. In the latter district, known as the "Telluride Belt," we have indisputable evidence of vein-impregnation by the circulation of siliceous waters through the joints of the porphyry. In this case, tellurium is the chief mineralizing agent of the gold.

Perhaps the most striking example of the deposition of gold through the agency of thermal waters may be seen in the celebrated Bassick mine, which has so often been described. This deposit exhibits all the characteristics usually accompanying geyser-action. The disintegration and destruction of the porphyry, with a partial replacement of the feldspar by silica, is here clearly shown. The gold was combined, almost in its entirety, with tellurium, and rich tellurides of gold and silver were found sprinkled through the mass of material. In some remarkable specimens, these tellurides formed a distinct coating on the surface of smooth boulders which had become rounded by attrition from the action of steam.

The notable deposits at Ouray, in the San Juan district, discovered during the last year, are worthy of mention in connection with this subject. These deposits are in quartzite, lying underneath an enormous flow of porphyry which covers the whole country in that district. The ore is associated with silica, mixed with an ochreous material which carries a large amount of sesquisulphate of iron. In every case where I have had an opportunity of examining samples of this ore, I have found bismuth in well-marked quantities, and, in one instance, I was able to distinguish crystals of one of the tellurides of bismuth, probably "wehrlite."

With a view of determining the results to be obtained by fusing

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pure iron pyrites with pure gold and some of its alloys, at a high temperature, I have made a number of experiments, the general import of which may be summed up as follows: Pure gold, melted with pure iron pyrites, is not attacked in any way by the fused sulphide of iron. A matte is obtained, in which the gold exists in a pure state, but disseminated throughout the mass in very fine globules.

If this experiment be repeated with equal weights of gold and silver, we find a very different state of things, for on grinding and submitting to the closest microscopic examination, no evidence whatever of gold, or its alloy with silver can be found. But on dissolving the finely-powdered material in nitric acid, a separation of a dark bronze crystalline powder is effected, which powder has been found to be composed of 89.06 per cent. gold and 10.94 per cent. silver.

It would appear from this experiment, that the presence of silver materially helps the solution of the gold in the matte; and possibly a sulphide of gold and silver is formed, which becomes decomposed by nitric acid, any excess of silver which may be present going into solution.

A mixture of pure iron pyrites with gold and bismuth gave, after melting, a matte which showed no traces of metallic gold; and on treating with nitric acid, a bronze-colored crystalline sponge was produced, which was found to contain 89.06 per cent. gold and 10.94 per cent. bismuth (by difference).

In this experiment there is every reason to believe that the gold entered into combination with the bismuth; that the resulting alloy was completely dissolved in the fused matte; and that since no silver was present to displace the bismuth, the latter metal remained combined with the gold in such a proportion as to resist altogether the action of nitric acid.

This experiment was repeated under exactly similar conditions, with the exception that silver was added. In this case, before the addition of nitric acid, small specks of the silver-gold alloy were distinctly visible to the naked eye. The subsequent treatment by nitric acid resulted in the sulphide of iron going into solution, and the separation of a crystalline sponge, which proved to contain 82.2 per cent. gold and 17.8 per cent. silver.

In this experiment no compound of gold with bismuth was formed, owing to the presence of silver; and this result confirms all previous experiments in regard to the relative affinity of these metals for gold.

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It would appear, from the results of these experiments, that pure iron pyrites, in a state of fusion, has no solvent action on gold; the matte obtained holding the gold in mechanical suspension, rather than in chemical combination. In the presence of other metals, such as silver, copper, bismuth and others, the gold forms compounds with these metals, and passes readily into what might be termed complete solution.

These results may not have any particular bearing on the natural occurrence of gold in pyritous ores; for we can scarcely bring ourselves to believe the conditions which favored the deposition of gold in ores, to be fairly analogous to the experiments I have described.

These experiments may be interesting, however, from a metallurgical point of view, as explaining the true condition under which gold exists in mattes.

With a view of ascertaining to what extent the results of the above experiments have a bearing on the condition of gold in copper matte, as produced in the ordinary process of smelting, I have made a careful examination of a sample taken from a large tap, weighing about four tons. This matte is a regular product of the establishment with which I am connected, and assayed, in this instance, 17 ounces of gold, 80 ounces of silver, and 65 per cent. of copper. It contained, also, such impurities as lead, bismuth, antimony and arsenic. The pulverized matte was treated with HNO_3 until it was entirely decomposed. After carefully washing by decantation, a small residue was obtained, which, under the microscope, revealed the presence of a bronze-colored crystalline alloy of gold and copper.

This result confirmed in a remarkable way those of all my previous experiments, viz., that, in the presence of copper, a definite alloy with *copper* is formed, while all the other metals, such as silver, lead, bismuth, etc., pass into solution in the matte, and in a condition to be dissolved or decomposed by nitric acid, and not, as is generally supposed, in combination with sulphur.

In one case I distinctly observed very minute octahedra of this gold and copper alloy.

In conclusion, I may be permitted to express a desire that the subject of this brief paper may be taken up by some one more competent than myself to do it justice. The cares and duties of a smelting-works do not leave much leisure for minute investigations of such a character, and I have only been able to skim the surface, and point out material for study. It would be interesting to extend the investigation of gold-ores to other localities, where the conditions

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may be somewhat different, to discover whether there is any general law which regulates the association of gold with other metals. The paragenesis of gold-bearing minerals would be a highly instructive subject for study, and I would take this opportunity of urging the members of the Institute, who are scattered all over the world, to contribute the results of their experience in regard to the occurrence of gold in their own particular districts.

The examination of specimens, collected in this way, might be taken up by the scientific department in Washington, which has already done much in the way of minute scientific investigation, as applied to mining and metallurgy. Work of such a character can only be conducted by specialists, who have all the advantages of a well-equipped laboratory at their command, and with whom time is of much less consideration than accuracy of results.

The United States Geological Survey is fortunate in having a staff of gentlemen, eminently qualified to conduct scientific investigations of this nature. There are many members of this Institute who, like myself, are engaged in the "science of making money out of ores" (as I believe Dr. Egleston has been pleased to define the science of metallurgy), who are obliged to fall back on the skilful chemist for aid in their work.

A true and earnest lover of his profession is ever on the alert, watching for new occurrences and new conditions, which will lead him to a more perfect knowledge of the laws which govern the behavior of metals and other elements, in their relation towards each other. At the present time there is, perhaps, no branch of metallurgical science more in need of such aid than the metallurgy of the precious metals.

The *Transactions* of the Institute contain many excellent examples of original work in this direction, but we require still more light of such a nature as that which has been recently shed on the treatment of gold-ores by chlorination, in the valuable paper on "The Losses in Roasting Gold-ores and the Volatility of Gold" (*Trans.*, xvii., 9), by Professor Christy, of the University of California.

I refer to this paper, not only for its scientific value, but also for its practical bearing on every-day work, and as an example which might well be followed by the heads of the scientific departments of other universities, scattered over the country.

Since my return from the Washington meeting I have made some further investigations in regard to the occurrence of tellurium in the

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Leadville pyrites, and I have found the most conclusive evidences of the existence of tellurium and bismuth in the pyrite from another mine in that district.

From a careful microscopical examination of some pieces of the ore, I can say without a doubt that the tellurium-bismuth mineral is the result of some secondary deposition by segregation; for I find abundant evidences of a grayish mineral, coating the surfaces of crystals of what appears to be pure crystallized pyrite.

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*PROGRESS OF METALLURGICAL SCIENCE IN THE WEST.**

BY RICHARD PEARCE, ARGO, COLO.

(Presidential Address at the Colorado Meeting, June, 1889.)

Discussion.

H. M. Howe, Boston, Mass. (Communication to the Secretary): One by one our venerated idols are shattered, our cherished beliefs are filched away. Mr. Pearce's interesting experiments on the effect of an intermixture of coal in expelling arsenic from roasting copper-ores seems at the first blush to spike one of the great cannons of our faith. Plattner, however, is so positive that coal does lead to the expulsion of arsenic† that we may well pause before accepting Mr. Pearce's results as final, and ask whether there is not some explanation of the discrepancy between them and Plattner's statements.

Plattner states that while ferric arseniate decomposes easily under the action of carbonaceous matter with expulsion of arsenic as arsenious oxide (As_2O_3) and sub-oxide of arsenic (it is now generally thought, I believe, that this supposed sub-oxide is really but a mixture of metallic arsenic and arsenious oxide); cupric arseniate, on the other hand, undergoes this decomposition with difficulty, usually forming a certain proportion of arsenide of copper.

Now, may not the whole matter lie just here? Mr. Pearce experimented on an ore containing its arsenic as enargite, sulph-arsenide of copper. This, in roasting, would probably give rise to arseniate

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* See page 55.

† Cf. Fetis's French translation, *Traité Théorique des procédés Métallurgique de Grillage, par Carl-Friedrich Plattner.* Noblet, Paris et Liège, p. 259 *et seq.*

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of copper, which, according to Plattner, is but slightly acted on by carbonaceous matter. May it not be that, if an ore containing arsenide of iron, say arseno pyrite, or an arsenical iron-copper matte had been used, part at least of the arseniate formed would have been arseniate of iron, and that this would have been decomposed by the coal, with expulsion of its arsenic?

MR. PEARCE (Communication to the Secretary): My experiments and conclusions refer exclusively to the roasting of *copper ores* containing arsenic, and not to compounds of iron and arsenic, and I cannot see that they conflict with the statements of Plattner, as given by Mr. Howe. The sample of ore I experimented upon, as stated in my paper, contained enargite, and I presume the conditions were not favorable for the formation of iron arseniate; in all probability the arsenic acid was combined almost entirely with oxide of copper.

My object in drawing attention to this matter was to demonstrate that the addition of coal at the end of the roasting operation of arsenical-copper ores did not appear to play the part which is generally claimed for it.

PHOSPHORUS IN CAST-IRON.

BY W. J. KERP, DETROIT, MICH.

(Ottawa Meeting, October, 1889.)

THOSE who have observed the influence of various elements upon cast-iron will be interested in the methods used by us to form the several series of test-bars, which form the basis of the conclusions presented in this paper.

We found that we could not, by adding ordinary white stick-phosphorus to molten iron, produce bars which would exhibit to the eye the peculiar influence of phosphorus.

By melting wrought-iron drillings with red phosphorus, Mr. H. S. Fleming produced in his laboratory 10 pounds of phosphide of iron, containing 10.22 per cent. of phosphorus. With this we made two series of test-bars, the first with our white-base iron, and the second with our F^{L}M gray-base iron. Before the time of making these bars we had been unable to procure regularly made pig-iron which contained enough phosphorus to permit of the production of

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a satisfactory series. In Vol. V. of the *Ohio Geological Reports*, and in various papers read before this Institute, a number of high-phosphorus irons are mentioned. Similar irons have been referred to, but all samples seem to have been destroyed. During the summer of 1888, Mr. Edward Orton, Jr., sent a man to Moxahala furnace in search of such iron, and he dug up two samples with 4.71 per cent. of phosphorus and 3.45 per cent. of silicon. Just before this we had procured a 2-pound sample of Hamden charcoal iron, made in 1854. Prof. Locke's analysis of this sample gave: graphite, 1.80; combined carbon, 0.50; phosphorus, 4.22, and silicon, 1.93.

Through the kindness of the owners of the Hamden property we were presented with a car-load of ore thought to contain 6.90 per cent. of phosphoric acid, delivered at Eliza furnace. As this furnace was just blowing out, the managers put this ore through on the last cast, and presented us with 1 ton of iron, containing 4.59 per cent. phosphorus, and 3.16 per cent. silicon (analysis by Mr. Orton). As there was not time to calcine the ore, it was put through raw, which caused the iron to be closer in grain than it would otherwise have been. The iron from Moxahala was made from a similar ore after calcination, and as the grain of this is almost exactly like that of the Hamden iron, we may feel that we could not have expected better results. From this iron, with our $F^{L}M$ base, we have made the Hamden series of this paper.

Mr. B. F. Fackenthal, Jr., superintendent of Durham furnace, sent us the last of samples saved from some experimental casts which they had made for the basic steel process. We selected the sample from his cast No. 7, phosphorus 3.57 (white), silicon 0.06, and made a test-bar from it, and then by mixing casts 7 and 15 (which had phosphorus 3.52, and silicon 0.32) we procured a white iron with phosphorus a little above 3.50, and silicon 0.186, which is the silicon of our white-base iron. With this mixture and our white base we made the Durham series of this paper.

For a more delicate test of phosphorus we requested the Stewart Iron Company to make for us a Bessemer iron containing 2 per cent. of silicon, and we received from them an iron containing 1.99 per cent. silicon, 0.092 per cent. phosphorus, manganese 0.5 and sulphur 0.15. The managers of Norway furnace sent us an iron containing 2.03 silicon, 1.65 phosphorus, manganese 0.87, sulphur 0.01, combined carbon 0.63, graphite 3.12. The pig that we used of each iron was the one from which drillings were taken for analysis, and each iron is of nearly the same constitution, except

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as to phosphorus, and of this element Stewart has less and Norway more than would be found in ordinary castings, and therefore a mixture of these irons gives us the Stewart-Norway series, corresponding nearly with metal used in ordinary foundry-practice.

We thus have five series from which to formulate conclusions regarding the influence of phosphorus in cast-iron; and as each series is made on an independent plan, we may place confidence in conclusions based upon them all.

The graphic representations of this paper are made on the same plan as those of former papers,* and having been considered in detail already, they need not be described here. All records are those obtained by "Keep's Tests." A little study of our tables will render them perfectly clear. We wish to repeat here that in all tests consideration is to be given, not to individual variations, but only to the general tendency of a whole series.

I. The Influence of Phosphorus upon the Grain of Cast-Iron.

Phosphorus produces a most peculiar grain, and of such a character that when once observed it is always readily recognized.

The 10.22 per cent. phosphide which is, as nearly as possible, iron and phosphorus, has a peculiar flat fracture, with each grain standing alone, and appearing as though it could be separated easily from those next to it. The color is very different from that of the fracture of wrought-iron or steel. It is almost white, with a tendency to straw-color. If broken when hot, the fracture of the castings, as they came from the crucible after the original mixture of red phosphorus and wrought-iron, was beautifully iridescent, shading from brilliant blues to bright gold. In many places there are cavities filled with fine needles of rich colors. In the fracture of the test-bars the same, though larger, bundles occur. The sides of these needles are brilliantly polished and straw-colored.

In other parts, where the metal is more solid, each grain is seemingly independent of the others. It is no doubt this peculiarity in the crystallization of iron and phosphorus which causes the weakness of high-phosphorus irons.

White and straw seem to be the colors imparted by phosphorus. In all further examinations of grain this peculiarity of phosphorus as to color must be remembered. In the Hamden iron the ground-

* *Trans.*, xvii., 683 and xviii., 102.

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work of the face of a fracture is white, with a slight yellow tint, and apparently partly imbedded in the surface are round grains, either intensely black and shining like glass beads, or else reflecting light so as to have such an appearance. The fracture of our gray F^LM base exhibited a dark, spongy grain. The grain of each individual member is such, that the grain of a series gradually changed from one of these extremes to the other.

By rubbing the finger across the fracture of the last member of the series, the grains do not cling to each other or to the finger, as in a strong, tenacious iron, but to some extent can be rubbed off.

The phosphorus exerts no influence to change combined carbon to graphite or *vice versa*. The graphite, if present, must be there through the influence exerted by some other element. In an iron deficient in graphite, or containing practically all of its carbon in the combined state, phosphorus has but little effect upon the grain. Iron so deficient in silicon that the carbon is almost wholly combined tends to a lamellar fracture. Judging from the Durham samples of white iron, we are led to think that phosphorus increases this tendency to a lamellar fracture.

Dr. Lord* describes an iron almost exactly corresponding to Durham, made from Hallelujah ore at Mt. Vernon furnace, Ohio, as having this lamellar fracture. To produce it, silicon must be exceedingly low. As the same fracture in flat specular surfaces is caused by other elements when silicon is low, the lamellar appearance is no indication of high phosphorus.

Incidentally we may mention in this connection that in irons even as low as from 1 to 2 per cent. of phosphorus, the surface of castings rapidly cooled is covered with a brassy yellow tinge. Often the face that runs against the chill is yellow, or the surface for three-quarters of an inch from the chill will be partly yellow.

Particularly will the fluid strip at the end away from the gate show this yellow color; with irons containing about 2¼ per cent. of silicon and 1 per cent. of phosphorus, their castings will shade on the surface from a blue to a golden tinge, which in some cases can be brushed off with the hand.

Tables I., II. and III. illustrate the appearance of the grain of the test-bars of the several series to which we have referred.

In forming each of the six series shown in Tables I. and II. we used, for melting the iron, a No. 16 Dixon crucible. In order to

* *Trans.*, xii., 506.

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Table I.

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insure as uniform a composition as possible for the whole series, we would have preferred to melt the whole 40 pounds of iron used, at once, adding phosphorus just before each cast. But as we had not facilities for handling so much iron, we melted 30 pounds of the base-iron at first and then made a 5-pound cast.

We then put back the runners of this cast, added about 2 pounds of the base-iron and enough of the phosphide (10.22 per cent.) to make the phosphorus in the cast of the desired percentage. We then made the next cast and so on. We kept the iron as hot as possible during the whole series.

The following questions presented themselves: 1st. What effect would this manipulation of returning sprues to be remelted and the long-continued heat exert on the test-bars? 2d. As the phosphide consists of 90 per cent. pure iron, what would be the effect due to the introduction of this iron without the phosphorus?

To answer these questions we made a series of the base alone, shown at the top of Tables I. and II., and another, shown in the center of these tables, exactly like it, to which we added an amount of iron wire equal to the quantity of pure iron introduced with the phosphide in the series shown at the bottom of these tables.

The influence of the manipulation of the iron is shown in the white-base iron series at the top of the table, and consists of the pictures of the fracture of test-bars one foot long and one-half inch square, broken in the middle. The square picture just above the test number represents the appearance of the fracture at the center of the bar; the long picture just above this shows the end of the bar which was cast against a chill, the side being split off to show the depth of chill, or the effect of sudden cooling.* With silicon as low as 0.25 per cent. it is almost impossible to obtain sound castings. The silicon in 376 and 381 was found by analysis, and the intermediate silicons were estimated.

It is therefore shown that by the manipulation, silicon was decreased at each cast and that the increased size of blowholes was due to this cause. As a further proof of this, see the engraving of the test of white iron 363, shown in our paper on Aluminum in Cast Iron, read at the Colorado meeting (*Transactions*, xviii., 110, Table II.). In this cast the silicon-content was 0.08 per cent. and the blowholes were in proportion to those of this series.

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* This series, as well as the upper series in Table II. was partially analyzed by Dr. C. F. Mabery, of the Case School of Applied Sciences.

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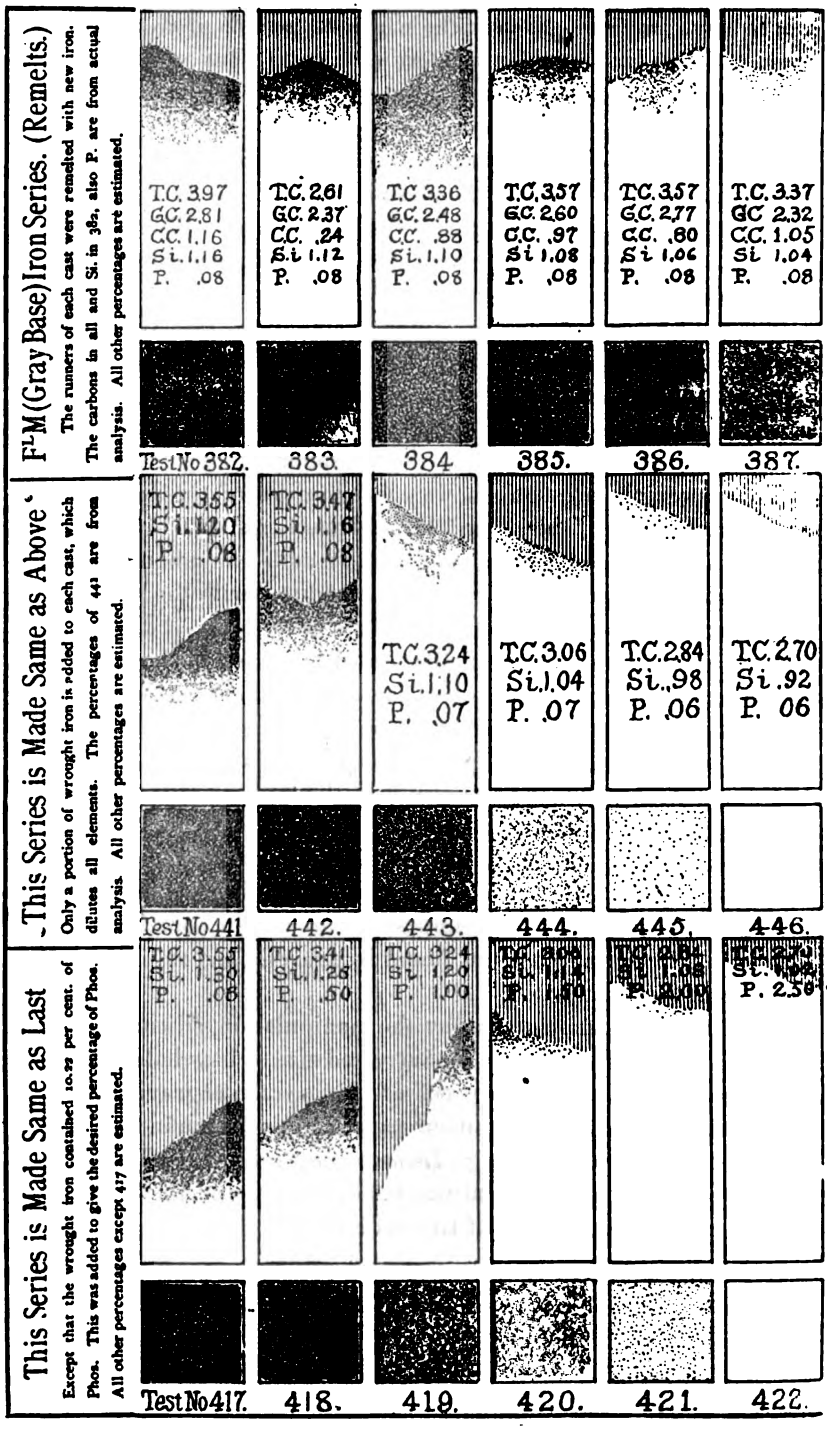
Table II.



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The effect of the addition of iron wire, shown in the center of the table, was to further decrease the silicon, which, as expected, increased the number and size of blowholes. While in the first series the carbons remained substantially uniform, the second series, in which wire was added *ad libitum*, decreased in carbon and phosphorus. In 440 the iron became so sluggish that we could not carry the series farther.

At the bottom of Table I. we have the phosphide series, which is exactly like the last, except that the phosphorus-contents steadily increase. The blowholes in this series do not increase, and are less in number than in the upper series. It proves that phosphorus lessens the tendency to form blowholes, because the iron remains fluid longer.

We have a further proof of this. The bars made from cast No. 7 of Durham, contained only 0.06 silicon and 3.57 phosphorus and made perfectly solid castings. Each number of the Durham series was perfectly solid and would have made a picture like the one on the right of 440. If it had not been for the phosphorus, the low silicon would have caused the castings to be very porous.

In Table II. the upper, middle and lower series correspond in each detail, so far as mixture and manipulation are concerned, with the series of Table I., except that instead of the white base used before, in this table the F^{L}M gray base is used. F^{L}M was not exactly uniform in silicon, as will be noticed in tests 382, 441 and 417. In each series, taken as a whole, silicon decreases, and as a consequence the depth of chill increases with each melt.

This is due to the decrease in graphite and the increase in the percentage of combined carbon caused by the decrease of silicon. The same causes lighten the grain of the fracture.

In the series at the center of the table, to which wire was added, the further decrease of silicon due to this increase of the iron caused the chill to increase in depth more rapidly, and the color to lighten more than in the first series. This effect is increased, since 441 is darker than 382 and yet 446 is perfectly white. In the last series in Table II. we began with a still darker F^{L}M iron, 417; and 421 is darker than 445. In this last series phosphorus has increased to 2.50 per cent., but allowing for the variation in silicon, this and the series above it are exactly alike, showing that the phosphorus had not the slightest influence on the color of the grain except the peculiar formation before spoken of.

In Table III. the first 11 tests are from a mixture of the F^{L}M base and Hamden pig-iron; 467 is Moxahala pig. As F^{L}M contained only 1.25 silicon, we added to each cast enough of a 14.77

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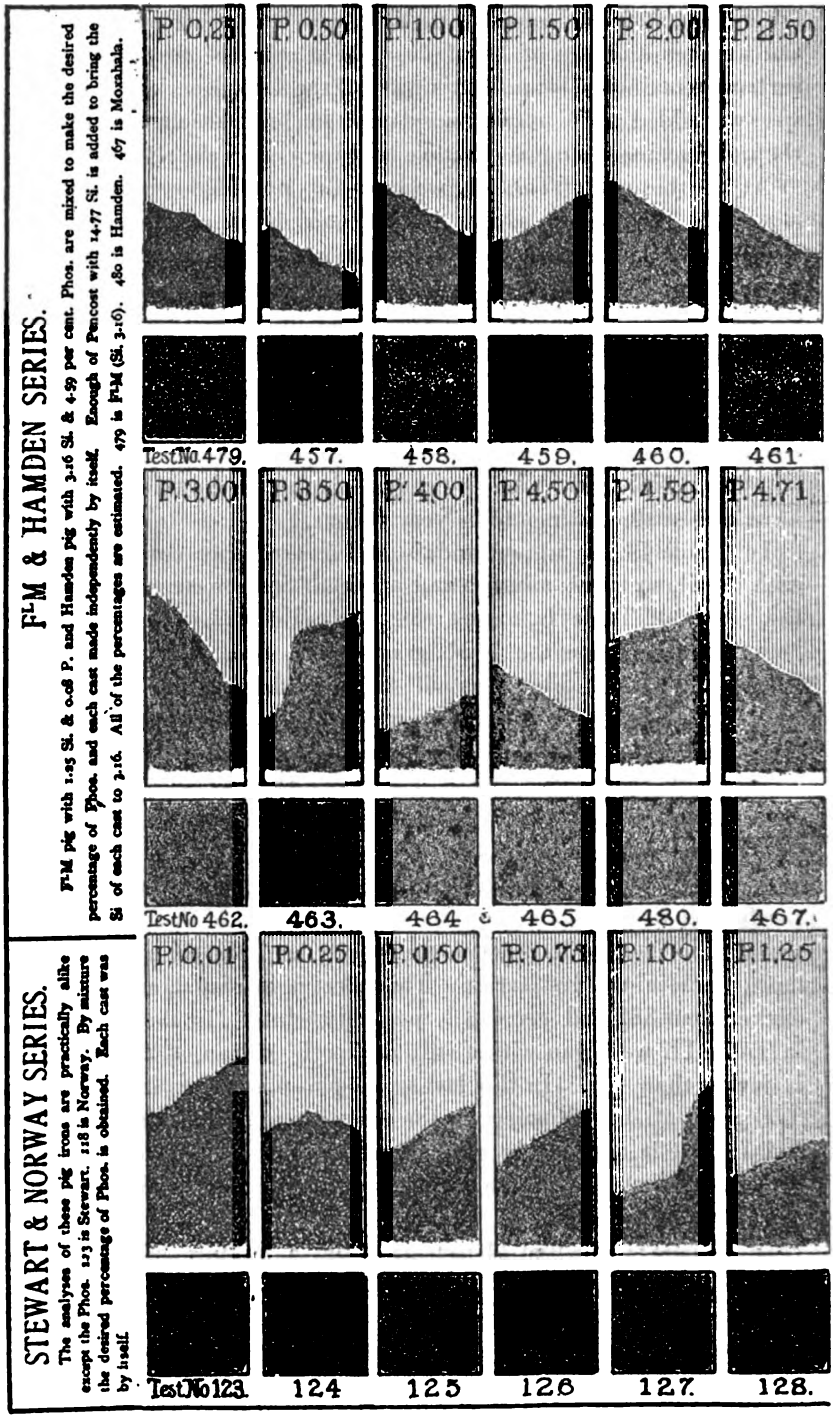
Table III.



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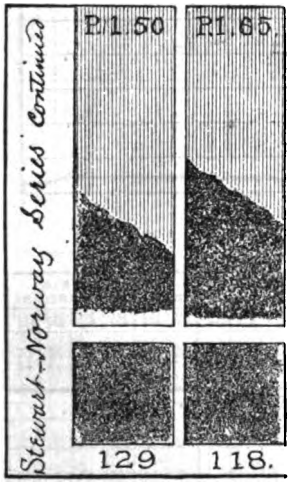
per cent. ferro-silicon to keep the silicon of each set of test-bars up to 3.16, which was the percentage of silicon in the Hamden pig-iron. In cast 479, which is of $F^{L}M$, with its silicon brought to 3.16 per cent., the chill is only a little over 0.10 of an inch deep. The chill is only slightly more in the case of 480, which is of Hamden alone, with 3.16 per cent. of silicon.

The chill of each cast of the series does not materially vary from that of the others; 467 is a cast of Moxahala pig-iron which was made from a different ore, and where the furnace was running regularly. It shows less chill than Hamden iron, while its phosphorus is higher.

In color, all phosphorus-iron is light. Hamden 480 is light, and Moxahala is still lighter, and both have the round, black, bead-like grains. In each phosphorus-series the color lightens with each addition of phosphorus. This series is perfectly satisfactory; the only thing to be regretted is that the silicon had to be doctored.

The last series in this table is free from all the objections to the previous series. The base-irons are Stewart and Norway pig-irons, and the composition of both is substantially the same, except the phosphorus. There is practically no phosphorus in the Stewart iron, while it is 1.65 per cent. in the Norway. The color varies as before and the chill remains the same—in fact, it decreases as phosphorus increases. As it is combined carbon that causes chill, we repeat that these tests show conclusively that phosphorus does not increase or decrease combined carbon. A conclusive proof of this has just come to us. We were requested to test a pig of No. 1 Star iron, and to furnish a chemical analysis. The physical test showed almost no chill, it being only 0.01 of an inch, or a mere film.

TABLE III*a*.



The report of the analyst, Mr. J. Blodget Britton, under date of June 27, 1889, is as follows: "Pure iron, 89.480; graphitic carbon, 3.405; silicon, 4.387; phosphorus, 1.470; sulphur, a trace; combined carbon and undetermined loss, 0.071." He says: "I failed in the attempt to determine the combined carbon. I found that, practically, strictly combined carbon was not present, or at least not in quantity to exert any influence." The remelting in the cru-

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cible usually causes some of the graphite of the pig to assume the combined form, but the slight indication of chill of this iron shows that in the pig combined carbon was wholly absent. The color of this iron is the same as that of the 1.50 per cent. phosphorus-tests of our series, and the grain is of the same peculiar kind.

As combined carbon is the thing to be avoided if soft castings are required, all elements that encourage the formation of combined carbon must be kept low. Phosphorus seems to be a perfectly passive element so far as carbon is concerned. As we proceed we shall see that phosphorus is a lessener of shrinkage, and that it prolongs the period of fluidity, and that if it were not for its weakening effect it would not be necessary to keep it at less than 1 or 1½ per cent. in a foundry-mixture.

II. The Influence of Phosphorus upon the Chill of Cast-Iron.

We have fully treated the subject of chill in the remarks just made, and introduce here the tables showing chill.

Table IV.

MIXTURES	FLM AND WROUGHT IRON + P. 10.23						FLM + SI (3.16) & HAMDEN PHOS. PIG IRON (P. 4.59 SI 3.16)											
TEST NUMBERS	417	418	419	420	421	422	479	457	458	459	460	461	462	463	464	465	480	467
Percentage of Phosphorus by Mixture.	.08	.50	1.00	1.50	2.00	2.50	.25	.50	1.00	1.50	2.00	2.50	3.00	3.50	4.00	4.50	4.59	4.71

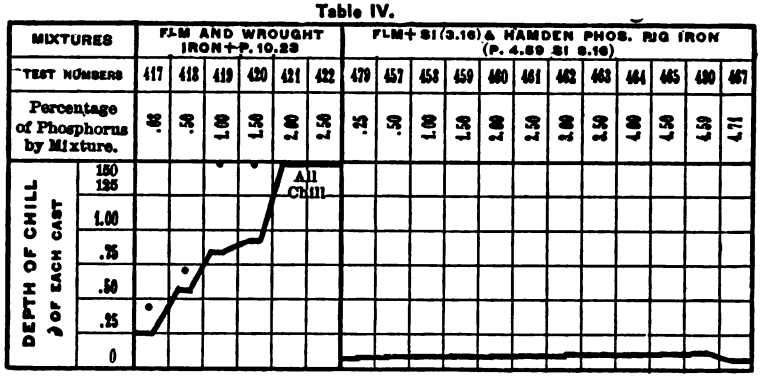
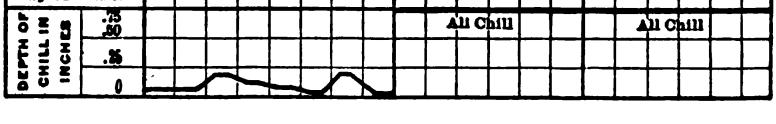


Table V.

MIXTURES	STEWART & NORWAY PIG IRONS								WHITE (BASE) & WRO'T FE. + P. 10.23						WHITE (BASE) & DURHAM PIG IRONS					
TEST NUMBERS	123	124	125	126	127	128	129	118	412	413	414	415	416	416²	363	484	483	482	481	481²
Percentage of Phosphorus by Mixture.	.01	.25	.50	.75	1.00	1.25	1.50	1.65	.25	.50	1.00	1.50	2.00	2.50	.25	.50	1.50	2.50	3.50	3.57



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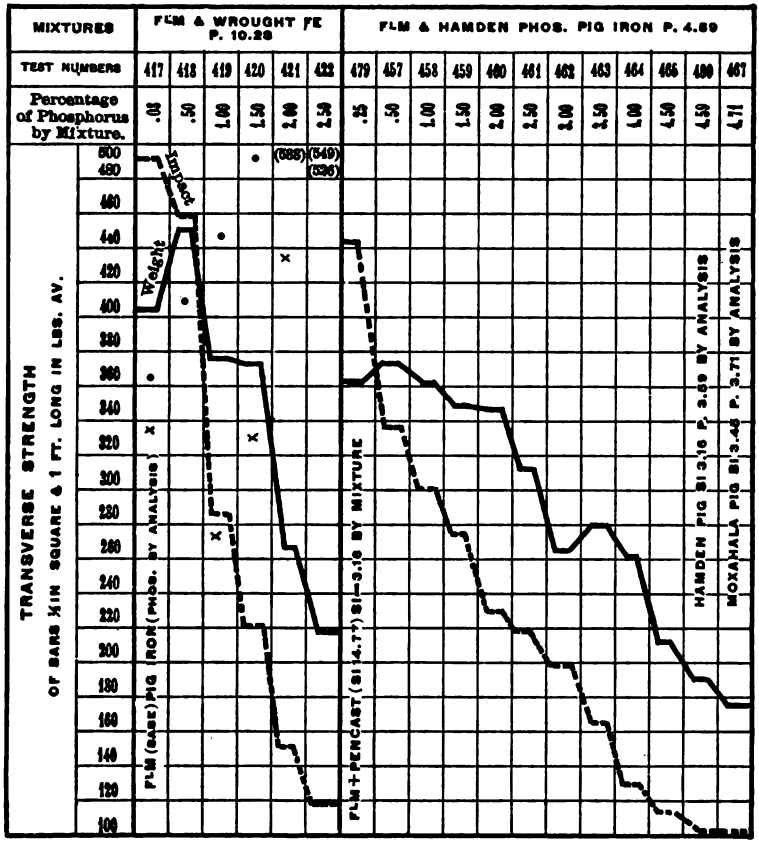
III. The Influence of Phosphorus upon the Strength of Cast-Iron.

Tables VI. and VII. show that while phosphorus of itself, in whatever quantity present, weakens cast-iron, yet in quantities less than 1.5 per cent. its influence in this direction is not sufficiently great to overbalance other beneficial effects, which are exerted before the percentage reaches 1 per cent. Probably no element, of

Table VI.



itself, weakens cast-iron as much as phosphorus, especially when present in large quantities.

In Table VI. of F^L M and wrought-iron with phosphorus we have recorded the transverse breaking-weight of the same mixture without phosphorus by dots (.). In the last two columns, the figures in parentheses at the top denote the breaking-weights. Impacts are shown

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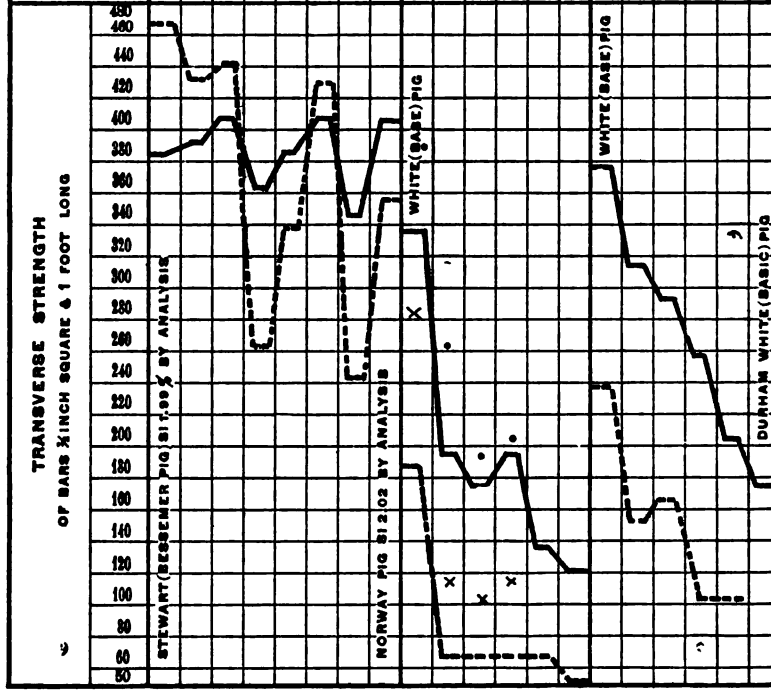
by a cross (x); the last record being in figures below the record of weight. The difference in the results with and without phosphorus are noticeable; and yet, judging from the other tables, we are led to believe that the variation shown by the phosphide series is all that should be ascribed to phosphorus.

An examination of the Hamden and Durham series, Tables VIII.

Table VII.

MIXTURES	STEWART & NORWAY (PA.) PIG IRONS								WHITE (BASE) & W'T FE (P. 10.23)						WHITE (BASE) & DURHAM (BASIC)					
TEST NUMBERS	123	124	125	126	127	128	129	118	412	413	414	415	416	416'	363	484	483	482	481	481'
Percentage of Phosphorus by Mixture.	.01	.25	.50	.75	1.00	1.25	1.50	1.62	.25	.50	1.00	1.50	2.00	2.50	.25	.50	1.50	2.00	2.50	3.57



and IX., shows that as phosphorus increases, the iron becomes weaker and the deflection is less. The iron is not, however, made to bend less by the introduction of phosphorus, as is shown by calculating the ratio of the deflection to weight. A graphic representation of these ratios would show that as soon as phosphorus has exerted all of the influence that it is capable of exerting, the iron becomes no stiffer by further additions. In the two phosphide series, with the F^L M base,

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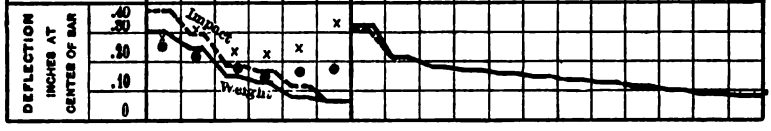
the iron becomes slightly stiffer as phosphorus increases, but this is because both carbon and silicon have been reduced by introducing pure iron in the effort to carry the phosphorus contents higher. In this series, with the white base, the beginning was made with too little silicon to influence the iron, and the additions of wrought-iron and phosphorus increased only the latter. A calculation of the ratios shows that the deflection increased and as the only change has been an increase of phosphorus, the conclusion is that phosphorus increases deflection, that is, it makes iron bend more in proportion to the weight borne before breaking.

Table VIII.

MIXTURES	FLM AND WROUGHT IRON + P. 10.23						FLM + SI. (3.16) AND HAMDEN PHOS. PIG (P. 4.59 SI. 3.16)										
TEST NUMBERS	417	418	419	420	421	422	479	457	458	459	460	461	462	463	464	465	480	467
Percentage of Phosphorus by Mixture.	.08	.50	1.00	1.50	2.00	2.50	.25	.50	1.00	1.50	2.00	2.50	3.00	3.50	4.00	4.50	4.59	4.71

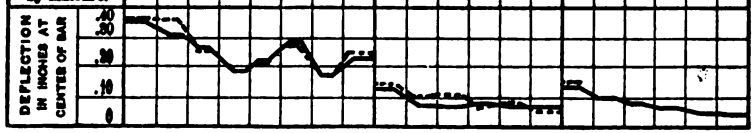


In Tables X. and XI. we present the records of the sets of the test-bars mentioned in this paper. It will be seen that the permanent set decreases as phosphorus increases.

Shrinkage is decreased when phosphorus is added, as is shown in Tables XII. and XIII.

Table IX.

MIXTURES	STEWART & NORWAY PIG IRON							WHITE (BASE) & WRO'T FE + (P. 10.23)						WHITE (BASE) & DURHAM PIG IRON					
TEST NUMBERS	123	124	125	126	127	128	129	118	412	413	414	415	416	416[2]	363	434	433	432	431	431[2]
Percentage of Phosphorus by Mixture.	.01	.25	.50	.75	1.00	1.25	1.50	1.65	.25	.50	1.00	1.50	2.00	2.50	.25	.50	1.50	2.50	3.50	3.57



In Table XII., from 417 to 422, dots indicate the shrinkage of the square bar in the series with wrought-iron added without an increase of phosphorus, and when the limit of the table is reached, numbers at the top of the column represent it. To find the influence due to the phosphorus, the difference between the iron series and the phosphorus series must be taken. This shows a great lessening of shrinkage by the phosphorus. In the Hamden series the shrinkage

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of $F^{L}M$ is very much lowered by the silicon added before cast 470 was made, but there is rapid fall in shrinkage until 1 per cent. of phosphorus is reached. The same is true for the Stewart-Norway series, though the change is not great.

Table X.

MIXTURES		FLM AND WROUGHT IRON + P. 10.23						FLM + SI. (3.16) & HAMDEN PHOS. PIG IRON (P. 3.59 SI. 3.16)										
TEST NUMBERS		417	418	419	420	421	422	479	457	458	459	460	461	462	463	464	465	480	467
Percentage of Phosphorus by Mixture.		.03	.50	1.00	1.50	2.00	2.50	.25	.50	1.00	1.50	2.00	2.50	3.00	3.50	4.00	4.50	4.59	4.71
PERMANENT SET IN INCHES	.12																		
	.08																		
	.04																		
	0																		

An iron very low in silicon should be taken to prove the effect of phosphorus on shrinkage, as in the last two series of Table XIII. In this case we begin with a very high and end with a comparatively low shrinkage. The decrease in shrinkage in phosphorus-irons cannot be ascribed to an increase in graphite, for we have proved that

Table XI.

MIXTURES		STEWART & NORWAY PIG IRONS								WHITE (BASE) & WRO'T FE. + P. 10.23						WHITE BASE & DURHAM PIG IRONS					
TEST NUMBERS		123	124	125	126	127	128	129	116	412	413	414	415	416	416[2]	369	484	485	482	481	481[2]
Percentage of Phosphorus by Mixture.		.01	.25	.50	.75	1.00	1.25	1.50	1.65	.25	.50	1.00	1.50	2.00	2.50	.25	.50	1.50	2.50	3.50	3.57
PERMANENT SET IN INCHES	.12																				
	.08																				
	.04																				
	0												No Set Perf		ect Spring						

phosphorus does not influence the carbon. It must be due to a direct action of phosphorus. To prove this more fully, we made a cast of as pure wrought-iron as we could find. The shrinkage was

Table XII.

MIXTURES		FLM AND WROUGHT IRON + P. 10.23						FLM + SI. (3.16) & HAMDEN PIG IRON (P. 4.59 SI. 3.16)										
TEST NUMBERS		417	418	419	420	421	422	470	457	458	459	460	461	462	463	464	465	480	467
Percentage of Phosphorus by Mixture.		.03	.50	1.00	1.50	2.00	2.50	.25	.50	1.00	1.50	2.00	2.50	3.00	3.50	4.00	4.50	4.59	4.71
SHRINKAGE IN. PER FOOT	.200																		
	.180																		
	.160																		
	.140																		
	.120																		

0.292 inch per foot. We then made a cast from our 10.22 per cent. phosphide, which was wrought-iron and phosphorus, analysis

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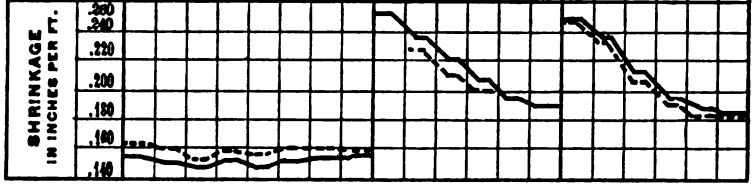
failing to reveal silicon or carbon. The shrinkage was 0.164 inch, so that over 40 per cent. of the shrinkage was taken out by the phosphorus. All high-phosphorus pig-irons have low shrinkage.

The further test of this phosphide of iron showed 58-pounds breaking weight with a deflection of .04 inch. Its fluidity was 7.

Table XIII.

MIXTURES	STEWART & NORWAY PIG IRON								WHITE (BASE) & WRO'T FE + (P. 10.23)						WHITE (BASE) & DURHAM PIG IRON					
TEST NUMBERS	123	124	125	126	127	128	129	118	412	413	414	415	416	416*	363	434	433	432	431	431*
Percentage of Phosphorus by Mixture.	.01	.25	.50	.75	1.00	1.25	1.50	1.65	.25	.50	1.00	1.50	2.00	2.50	.25	.50	1.50	2.50	3.50	3.67

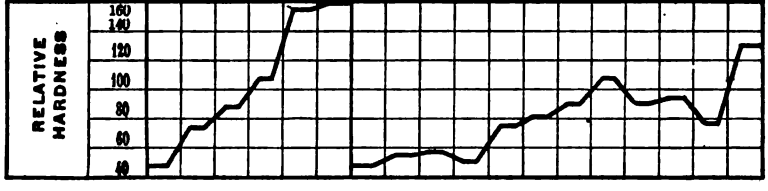


It had no chill and the fracture showed it to be a perfectly white iron.

From our knowledge of the effect of silicon through its action on carbon, and of phosphorus acting directly on the iron, the least possible shrinkage must be expected in iron with the largest amount of graphite and phosphorus contents to produce the maximum effect of the latter. This is the case. While we do not propose to discuss phosphorus in pig-iron in this paper because of the modifications

Table XIV.

MIXTURES	FLM AND WROUGHT IRON + P. 10.23						FLM + SI. (3.18) AND HAMDEN PHOS. PIG (P. 4.59 SI. 3.18)										
TEST NUMBERS	417	418	419	420	421	422	479	457	458	459	480	461	462	463	464	465	466	467
Percentage of Phosphorus by Mixture.	.03	.50	1.00	1.50	2.00	2.50	.25	.50	1.00	1.50	2.00	2.50	3.00	3.50	4.00	4.50	4.59	4.71



produced by other elements present, yet we may note that the shrinkage of Star iron, already alluded to, was only 0.140. If the phosphorus in this iron had been below 0.5 the shrinkage would probably have been 0.180.

In a former paper we have shown that the ordinary cause of hard-

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ness in cast-iron is combined carbon, and that under certain conditions silicon may exert a softening influence, but that excessive silicon causes hardness.* In this paper we propose to show that phosphorus does not ordinarily harden cast-iron and probably for the reason that it does not increase combined carbon. Further experiments might show exceptions, for our 10.22 phosphide is hard, while wrought-iron cast alone is soft. Hamden is hard, as also is Moxahala, but we are not prepared to state whether the hardness of the last two is from excessive phosphorus. The mixture of the hard Hamden with soft 479 shows hardness to increase as Hamden increases in quantity. In the first series in Table XIV. the increase of hardness is due to

Table XV.

MIXTURES	STEWART & NORWAY PIG IRONS								WHITE (BASE) & WRO'T FE. + P. 10.23						WHITE (BASE) & DURHAM PIG IRONS					
TEST NUMBERS	123	124	125	126	127	128	129	118	412	413	414	415	416	416²	363	484	483	482	481	481²
Percentage of Phosphorus by Mixture.	.01	.25	.50	.75	1.00	1.25	1.50	1.65	.25	.50	1.00	1.50	2.00	2.50	.25	.50	1.50	2.50	3.50	3.57
RELATIVE HARDNESS 140																				
120																				
100																				
80																				
60																				

the increase of combined carbon due to the manipulation and not to increased phosphorus. The hardness records of Table XV. indicate that phosphorus is neither a hardening or a softening agent, but if it is either, the tendency is probably on the side of hardening.

The conclusion reached from the tests recorded in Tables XVI. and XVII. is that the fluidity of the metal is slightly increased by

Table XVI.

MIXTURES	FLM AND WROUGHT IRON + P. 10.23						FLM + SI. (3.16) & HAMDEN PHOS. PIG IRON (P. 4.59 SI. 3.16)										
TEST NUMBERS	417	418	419	420	421	422	479	457	458	459	460	461	462	463	464	465	480	467
Percentage of Phosphorus by Mixture.	.00	.50	1.00	1.50	2.00	2.50	.25	.50	1.00	1.50	2.00	2.50	3.00	3.50	4.00	4.50	4.50	4.71
FLUIDITY OF EACH CAST IN INCHES 12																		
8																		
4																		

phosphorus, but not to any such great extent as has been ascribed to it. By watching the cooling of the metal in the mould after being poured, it was found that the time required for setting varies directly with the percentage of phosphorus, the phosphide cast holding its heat longer than any other. This property of remaining long in the fluid

* *Trans.*, xvii., 702.

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state must not be confounded with fluidity, for it is not the measure of its ability to make sharp castings, or to run into the very thin parts of a mould. Generally speaking, however, the statement is justified that, to some extent, phosphorus prolongs the fluidity of the iron while it is filling the mould.

General Remarks.—We have endeavored to exhibit the action of phosphorus separated from all other influences. When we understand what is to be expected from phosphorus we are able to detect its presence by its peculiarities. But we must not expect that a given percentage of phosphorus will behave at all times as it has done in our tests, for other elements may be present in such a way as to change the results entirely.

The peculiarities of these records naturally lead us to review the question from a practical standpoint. Are the favorite foundry-irons high in phosphorus? The old Scotch irons contained about 1

Table XVII.

MIXTURES	STEWART & NORWAY PIG IRONS								WHITE (BASE) & WRO'T FE. + P. 10.29						WHITE (BASE) & DURHAM PIG IRONS					
TEST NUMBERS	123	124	125	126	127	128	129	118	412	413	414	415	416	416[2]	363	484	483	482	481	481[2]
Percentage of Phosphorus by Mixture.	.01	.25	.50	.75	1.00	1.25	1.50	1.63	.25	.50	1.00	1.50	2.00	2.50	.25	.50	1.50	2.50	2.50	3.37
FLUIDITY OF EACH CAST IN IN. 12																				
8																				
4																				
0																				

per cent. of phosphorus. The foundry-irons which are most sought for for small and thin castings in the Eastern States contain, as a general thing, over 1 per cent. of phosphorus. From an extensive examination of English foundry-irons, Prof. Turner concludes that the best foundry-iron should contain about 1 per cent. It has always been noticeable that the irons which are rejected by the rolling-mill people on account of high phosphorus are most acceptable to the founder.

Certain irons which contain from 4 to 7 per cent. silicon have been so much used on account of their ability to soften other irons that they have come to be known as "softeners" and as lesseners of shrinkage. We have shown in a former paper that the silicon which they contain can only soften and lessen shrinkage through its power to change combined carbon to graphite and this only to a limited extent.* Otherwise silicon hardens cast-iron and increases shrinkage. These irons are valuable as carriers of silicon; but those accustomed to their use will perceive that the irons which are sold

* *Trans.*, xvii., 697, 702.

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most as softeners and shrinkage-lesseners are those containing from 1 to 2 per cent. of phosphorus. We must therefore ascribe the reputation of some of them largely to the phosphorus and not wholly to the silicon which they contain.

It is new to consider phosphorus in this light in foundry-practice, but it must be remembered that from one-half to 1 per cent. will do all that can be done in a beneficial way, and that all above that amount weakens the iron, without corresponding benefit. It is not necessary to search for phosphorus-irons. Most irons contain more than is needed and the care should be to keep it within limits.

ALUMINUM IN THE DRAWING-PRESS.

BY OBERLIN SMITH, BRIDGETON, N. J.

(Washington Meeting, February, 1890.)

THE experiments described in this paper are very incomplete and only preliminary to those I hope to make in the future. Having had a good deal of experience in cutting, forming and drawing sheet-metals, and feeling a great interest in the commercially new and wonderful metal, aluminum, I applied to my friend, Mr. A. E. Hunt, for some specimens in sheet-form, that I might, from time to time, test its working-properties in presses with which we happened to be experimenting at the works of the Ferracute Machine Company.

The chances for such trials being accidental only, and the time having been limited to the last few weeks, no crucial experiments have been tried, and no exact data have been obtained for a rigid comparison with other metals. The results of what little we have done, however, are shown in the articles exhibited in connection with the reading of this paper, which comprise various cups, plates, boxes, bells and cartridge-cups and shells. For the latter I am indebted to my friend, Mr. Hobbs, of the Union Metallic Cartridge Co., who, at my request, ran several pieces of sheet-aluminum through the dies with which he was making brass cartridges. The specimens are rougher than they would be had the metal not been a little too thin for the particular dies which he happened to be running.

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They show, however, a wonderful, and to me a somewhat unexpected, adaptability to this kind of work. Mr. Hobbs writes me as follows:

"The metal does not appear to stand nearly as much strain in bending as brass or 'gilding' (see primer-pockets in large cups), but in drawing will apparently stand much more than either, without annealing. The longest tubes sent you were brought to the present condition in three drawings without any annealing, the metal being .026 inch thick."

We thus see that a very valuable property of aluminum in this kind of work consists in its not needing the frequent annealings

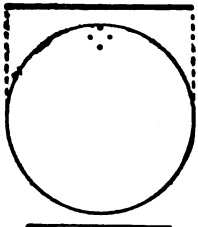


Fig. 2.

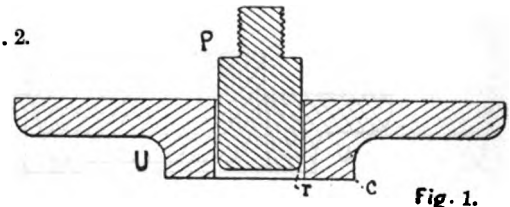


Fig. 1.

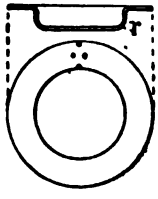
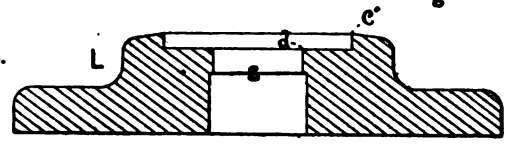


Fig. 3.



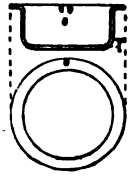


Fig. 4.

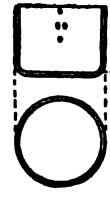


Fig. 5.

which are required by the brass in common use, and also by iron and steel, when drawn into deep articles. The failure in bending referred to is perhaps rather a failure to stretch, as shown by one of the samples, where the metal has been torn partly through, in attempting to draw a deep "pocket" or depression in the flat surface. This is probably accounted for by the known tendency of the metal to elongate under tension locally, rather than over a considerable space.

The stretching of a pocket of this kind does not, however, come under the "drawing-process" proper, which consists, not in stretch-

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ing the metal, but in causing it to flow together, or "upset," in a circumferential direction, while at the same time it is flowing further apart, or stretching, in a radial direction. Thus treated, its thickness remains the same, except in some cases, where it is subjected to a combined operation—"broaching" and "drawing"—as happens in some of the operations of cartridge-making, etc. This broaching consists in squeezing the metal thinner, by having the space between punch and die less than its original thickness.

As what is known as the drawing-process may not be familiar to all, I will briefly describe it, in its simplest form, as used for drawing a flat disk of sheet-metal into a cylindrical or cup-like

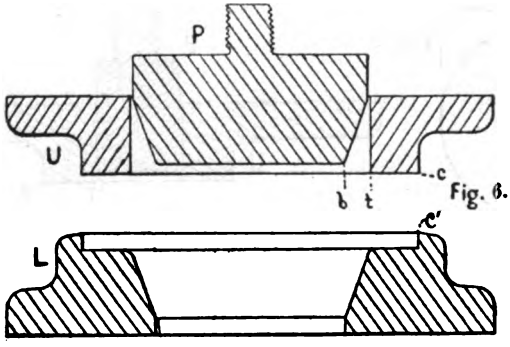

Fig. 6.

Fig. 7.

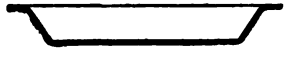

Fig. 8.

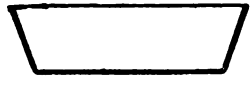

Fig. 9.

shape. The operation consists in holding this disk so tightly between the two parallel flat annular surfaces of the dies that it cannot wrinkle, while the punch which gives it its interior form is pushed down, through a hole in the upper die, into a recess in the lower die, which fits it with the proper amount of looseness to give space for the metal between. A set of these dies is shown in Fig. 1, in vertical section through their axis. In Fig. 2 is shown an axial section and a top view of the disk of metal, technically known as a "blank," before it is drawn. In Fig. 3 are shown the same views, respectively, at a later stage of the operation, when it has been drawn

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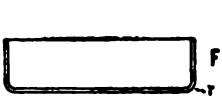
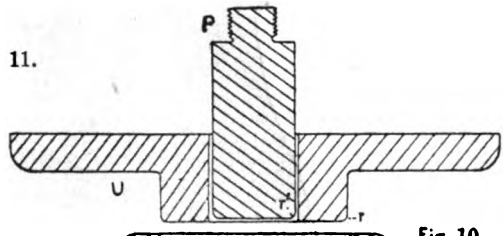
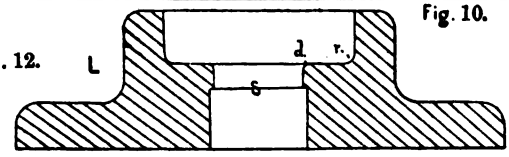
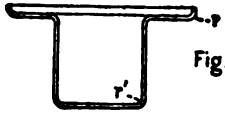
to about one-third of its final depth. In Fig. 4 it is shown at, say, two-thirds depth, and in Fig. 5 as completed. Four small dots will be noticed in Fig. 1, marked upon the blank, and forming the corners of a square, with its diagonal placed in a radial line. In the subsequent figures it will be noticed that two of these dots, upon either side of this imaginary radius, have gradually approached each other, while the other two dots, lying in the radial line, have receded, thus beautifully illustrating the respective directions in which the molecules of the metal have travelled in reaching their new locations. In Figs. 6, 7, 8 and 9 are shown a set of dies, and the work in its several stages, of a form conical rather than cylindrical.









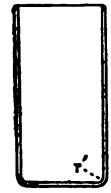


Fig. 14.

The dies as shown in Figs. 1 and 6 are made so that they cut the disk from a large sheet of metal at the cutting-edges, c and c', as the upper die, U, descends to perform its mission of holding the blank from wrinkling between its lower flat surface, and the corresponding flat surface of the lower die, L. When U has descended until it is firmly in contact with the blank, it stops and remains rigidly in position until the punch, P, has descended and drawn the metal from between the two holding-surfaces, over the rounded corner, d, and down below s, into its desired shape. The lower edge of the working-part of the die, s, is called the "stripping-edge," and is kept quite sharp,

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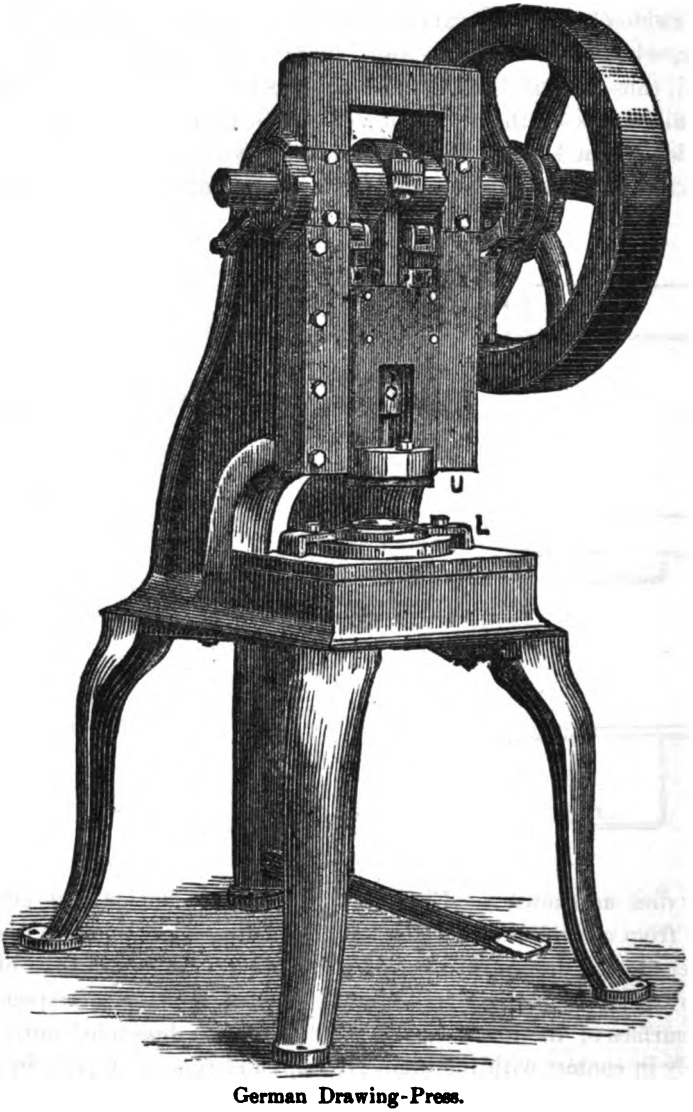
so that the metal will not be pulled up again by the friction of the punch. Although it has just passed through the die, its elasticity expands it very slightly, so that this stripping-edge is usually found

Fig. 15.



German Drawing-Press.

sufficient to keep it down. That a blank not confined by a holding-surface, will wrinkle, is a fact easily proved by experiment.

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In practice it is found necessary to put an ample vent-hole for air through the punch, or else the cup is pulled up afterward by suction, in spite of the stripping edge. In one of the sample cups shown, the vent-hole was insufficiently large for aluminum, although it had worked all right with tin-plate. This difference was owing to the stiffness of the latter material. The aluminum, being softer, was pulled up by the air-suction just before the punch left the cup. This, of course, would not have happened had the press been running at a slower speed. The dies shown will do equally well for blanks which have been previously cut elsewhere. In some cases, instead of the cutting-edges, *c*, *c*, a mere rim, with a rounded corner, projects above the surface, serving as a gauge to locate the blank centrally when it is thrown in.

In Fig. 10 is shown a pair of "deepening-dies," so-called, which take a cup, Fig. 11, that has already been drawn in dies, like Fig. 1, to as great a depth as the metal will stand, and deepen it to the form shown in Fig. 14. The successive intermediate stages of the operation are shown in Figs. 12 and 13. These and subsequent similar processes are the ones used in cartridge-drawing, except that the metal is usually somewhat thicker than the space between the punch and die, so that it is also subjected to a stretching process similar to wire-drawing. This reducing in thickness is often called "broaching," but there is considerable confusion of terms used by various manufacturers.

The presses used for all the dies just described are, of course, of special construction, being made with an "inner" and an "outer" ram. The latter carries the upper die and is arranged to stop for a time, after a part of the stroke has been made, while the inner ram, carrying the punch, descends to the bottom of its stroke and rapidly returns, just as does the ram in an ordinary single-action press. In Fig. 15 is shown a small drawing-press for work of this kind which will explain itself. It is a picture of a German press, and shows fairly well the principle involved. There are several kinds made in America, of better design, of which it might seem invidious to show one without the others.

A fuller account of this interesting process, together with rules for the construction of the tools used and a description of the products obtained, may be found in a lecture which I had the honor to deliver before the Franklin Institute, and which was published in its *Journal* for November, 1886.

In general, I can say, with regard to the drawing of aluminum,

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that, as far as my experiments have gone, it seems to be excellently adapted to be drawn into a great variety of household utensils, parts of scientific instruments, ornamental hardware, cartridge-shells and for very many other purposes, some of which we, probably, can scarcely conceive of as yet. One of its great advantages is, as before intimated, its capacity of working without frequent annealing. In the imperfect trials made at our works, in dies which were not especially prepared for it, it seemed to work (outside of the non-annealing) about the same as soft brass, but in some cases was not quite as tough. It, of course, did not show as well as tin-plate in regard to tensile strength. As far as I can now judge, it may be said, generally, that it is well adapted for deep drawing, and that the success of any given shape and proportions will, in comparing with other metals, be directly proportionate to tensile strengths. This view, however, may be somewhat modified in future, as the result of more careful and accurate experiments.

Since the foregoing paragraphs were written, I have seen some "drawn work" that Mr. Hunt had made, which is larger than any for which I happened to have dies on hand. Among his specimens are a sauce-pan and a tea-kettle, which corroborate my impressions as to the perfect adaptability of aluminum to the making of deep, seamless utensils,—which will open to the housekeeper a new era of healthfulness and cleanliness in the *cuisine*.

ALUMINUM IN SEARCH OF A NICKNAME.

BY OBERLIN SMITH, BRIDGETON, N. J.

(Washington Meeting, February, 1890.)

THE object of this paper is not so much reformatory, as historical and prophetic. History tells us that, for several months past, aluminum has, in one of its largest American manufactories, been freely nicknamed "Al" by everybody connected therewith, simply because its Christian name is too long for frequent speaking and writing. In another prominent factory, for the same reason, it is spoken of by its chemical symbol merely, pronouncing it "aye ell." These facts will, perhaps, explain the use of the final word in my title, which might otherwise seem undignified in a paper of this kind.

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In predicting a better future name for the noble and beautiful metal, which has so recently entered upon a practically commercial era, I use the word "nickname" in its best sense, viz., as expressing familiarity, not reproach.

In glancing over a list of what have heretofore been known as our commercial metals, we find that nearly all of them have short, crisp names which are words of one syllable, and these mostly of Anglo-Saxon derivation. The most notable exceptions, viz., silver, nickel and copper, are simple and easily-spoken words of two syllables. Did I aspire to be a professional humorist, I might say, that gold, tin, zinc, lead and the mixed metals, bronze, brass and steel, have attained their great success in life on account of their short and handy names. Soberly speaking, however, the converse idea is probably true that, on account of these metals being so generally useful, brief and convenient names were given to them, or came to them by a process of evolution.

It seems to me that, at some time in the future, our beautiful aluminum, which is perhaps destined to be more universally used than any other one metal, will need a *name* more in consonance with the rapid-going spirit of the time, and the same thing may be said of such other metals, with many-syllabled chemical names, as may come into frequent use. There is, of course, a good reason for much of the suggestive nomenclature of the chemists, inasmuch as their names not only give personality, but at the same time give a record of origin and family history. This is good for laboratory purposes, and, furthermore, either of the present names—aluminum or aluminium—is beautiful to the ear, and (unlike too many of our words) is properly spelled.

As intimated above, however, I think that, at some time in the future (possibly when either English or Volapük shall have become the world's language), we shall, for convenience, need a shorter name. It is, perhaps, premature now to make any suggestions as to what this will be. Possible words that occur to me, however, are "Aluum," "Alium," "Alim," and "Alm." These are all suggestive of the original word, but the last-mentioned is probably the best, as being the shortest and as appearing in print less like "Alum." It is to be hoped, however, that something more dignified than the one already started, viz, "Al" (pronounced like the nickname for "Albert"), will finally be adopted. There is no more reason why some such suitable name should not in the future be given to this metal than there is why it should not now have two different names. The reform in this matter has, perhaps, really begun—as many of us have

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decided to use the shorter name of the two; although, probably, rhetorically speaking, it is not as euphonious and dignified as is "aluminium." In witness of a feeling favorable to this reform may be mentioned the fact that one of our prominent engineering journals has recently declared in favor of "aluminum," simply because it is shorter, and has editorially expressed the intention of using it uniformly in the future.

Another reason for using the shorter word is that its oxide, alumina, has, by common usage, already been shortened from the word formerly used, "aluminia."

ALUMINUM BRONZE AND BRASS AS SUITABLE MATERIALS FOR PROPELLERS.

BY EUGENE H. COWLES, LOCKPORT, N. Y.

(Washington Meeting, February, 1890.)

Now that a determined public effort is being made on this side of the Atlantic to create a steel ship-building industry, the materials that are demanded for this purpose assume a new interest to the American metallurgist. In the past, the uses of metals in this country have been almost exclusively connected with the land, while in England their application to marine purposes has been equal in importance to that on shore. In the future the same should be the case here. This is the reason for bringing a somewhat nautical or rather naval topic before a body the chief interests of which are supposed to lie in the mines and mountains of the interior of the country.

A modern war-ship or "ocean-grayhound" (like the City of Paris) is an epitome of philosophy and human skill. In her graceful lines we have, as Ruskin might quaintly say, a "builded geometry;" while in her wonderful machinery for motion, offensive and defensive, we have the summation of the ideas of a host of students who have, at one time or another, during the course of a hundred years of her evolution, contributed their intellectual mites to render her a thing of beauty and of power.

Notwithstanding her present perfection, the modern ship contains room for further improvement. One of the most important things about her is the propeller. This active member is of more impor-

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tance than many constructors have realized or been willing to admit. Yet we have the most undoubted evidence that a slight change in its form, or in the material of which it is composed, or even in the condition of its surface, has made most decided changes in the speed of the ship and the size of her coal-bill.

With the form of the propeller the writer does not care to deal; that is meat for our mathematical friends, the designers. The question discussed in this paper is solely one of material; on this point there are data at hand of considerable value and interest.

Like the form of the screw, the material of which it is composed has been frequently changed. In the beginning it was made of cast-iron; succeeding this came the famous gun-metal or tin-bronze, in England a composition of 86 to 88 per cent. copper, from 2 to 5 per cent. zinc, and from 8 to 12 per cent. tin, while the American standard gun-bronze is composed of tin, 10 per cent.; zinc, 2 per cent.; and copper, 88 per cent. Following these materials came forged steel blades and phosphor-bronze, a new form of gun-bronze with a small amount of phosphorus added to flux it and harden it. Succeeding the last two, there appeared simultaneously propellers of mild steel cast in sand, and manganese-bronze, so called, but in reality a muntz metal with a composition of from 40 to 45 per cent. zinc and from 60 to 65 per cent. copper, with about as much ferromanganese as would replace the zinc burned off in the melting.

Following these came wheels made from Dick's delta-metal, an alloy similar to manganese-bronze, but with iron used instead of ferromanganese. And last on the list appear Cowles aluminum-bronze and aluminum-brass.

The materials from which propellers are made stand, therefore, in the following chronological order of their first use:

1. Cast-iron.
2. Gun-bronze.
3. Forged steel.
4. Phosphor-bronze.
5. Manganese-bronze.
6. Delta-metal.
7. Aluminum-bronze.
8. Aluminum-brass.

Cast-iron has proved too weak and corrodible for use in fast ships, and has largely fallen into disuse on sea-going vessels of the better class, although it is still universally used on the craft of the great

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lakes. Its weakness is such that blades of it have been known to be whipped off from large wheels in heavy seas by the racing of the engines. So subject is it to corrosion by salt water that Mr. W. C. Wallace stated, before the Institution of Naval Architects of England, that its average life is but five or six years. Yet it is agreed by eminent English authorities that cast-iron resists corrosion better than steel. Still worse characteristics of cast-iron are its utter lack of ductility, and its inability to withstand a shock, when used in thin masses.

Gun-bronze, upon being substituted for cast-iron, resisted successfully the corrosion by salt water, but it was found that the same trouble came up in a new form, namely, that the bronze wheel, when operated on a steel or iron ship, constituted a galvanic couple, the ship being one pole, the wheel the other, and that the action set up resulted in a serious pitting and corrosion of the iron plates in the stern of the ship. This new difficulty was happily overcome by the attachment of plates of zinc around the inside of the aperture in which the propeller works. The zinc, forming a new couple with the bronze, relieved the iron from the pitting action.

Another marked improvement seen by substituting gun-bronze for cast-iron was in the matter of speed. It was clearly proved that the same ship, with the same size and shape of screw, would make greater speed on less coal with a bronze screw than with one of iron. The causes were plainly discernible why this should be so: the bronze screw holds a finer edge in use than the iron, which becomes dull by corrosion, and can only be driven at a great waste of power and money. The bronze wheel keeps also a smoother surface, by reason of its non-corrodibility, and by thus saving friction makes the increase of speed and economy.

Had the world been content with the slow-going, low-powered ships of fifteen or twenty years ago, there would not have been much reason to ask for anything better than the old gun-metal. In its time and place it did good work.

The tensile strength of gun-bronze varies from 6 to 10 tons in actual practice. The writer has cut bars from bits of old wheels, furnished by our Navy Department, and the best result obtained from these was from 5 to 7 tons tensile strength. At the same time he has seen specially-prepared cast bars run as high as 25 tons in tensile strength, and with 20 per cent. of ductility. English authorities claim for it 12 to 16 tons tensile strength, and from 1 to 5 per cent. of ductility. These results were probably on small bars.

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On bars 2 inches in diameter and 15 inches between marks, Chief Engineer Harris, of the U. S. Navy, obtained a strength of but 10 to 12 tons per square inch and 3½ per cent. ductility.

To secure greater strength and stiffness, the English shipbuilders next tried forged-steel and cast-steel wheels, and many wheels of this kind are now in use. In them there is plenty of strength and toughness, but the question of corrosion reappears; and it is a most serious one. Mr. W. C. Wallace, a naval architect of high repute in Scotland, who has collected a vast amount of valuable data on this subject, states that he has come across many vessels, trading in different parts of the world, the steel wheels of which had not lasted more than three years. His testimony as to the very rapid corrosion of steel is sustained by Admiral Colomb, of the English Navy, by the experience of the firm of Messrs. Jessop & Co., as related by their Mr. Hall before the Institution of Naval Architects, and by Mr. G. W. Mannuel, Superintendent Engineer of the P. and O. Steam Navigation Co., and other distinguished engineers.

Phosphor-bronze was found to be little if any better than the gun-bronze in the matter of strength, although, of course, sufficiently non-corrodible. Its much higher price precluded an extensive use of it for propellers.

Delta-metal, like phosphor-bronze and manganese-bronze, came into a very limited use in the effort to secure a strong and yet non-corrodible material. It is not made in America, and we have little authentic information as to the success attained with it, although the proprietor, Mr. Dick, has informed me that large propellers have been successfully cast from it. When properly made, it is stronger than gun-bronze or manganese-bronze. It has about the same composition as the sterro-metal of General Uchatius, of Austria.

Manganese-bronze, so called, is in reality a manganese-brass, for zinc instead of tin is the chief element added to the copper. What its real strength and ductility is, few if any people know. Mr. P. M. Parsons, the proprietor of this brand of metal, claimed for it, in *London Engineering*, of April 20, 1888, in a controversy with the writer, a tensile strength of from 24 to 28 tons on small bars when cast in sand. He also claimed for it a transverse strength of 2 long tons on a bar 1 inch by 1 inch by 12 inches, and I do not doubt that such results have been obtained by him. Mr. W. C. Wallace states that brass-founders of high repute in England will only admit that manganese-bronze has from 12 to 17 tons tensile strength and 28 cwt. transverse strength. Mr. Horace See, recently head de-

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signer for the Messrs. Cramps, stated lately, in the New York *Times*, that he had privately tested the metal used in the U. S. ship Yorktown, and that it developed a registered tensile strength of 45,000 pounds per square inch, and from 6 per cent. to 12½ per cent. elongation. He did not state the dimensions of the bars, nor how they were made—two all-important considerations.

The Messrs. Cramp & Co. claim for the metal a strength of from 45,000 to 60,000 pounds, and I do not doubt that such results have occasionally been attained on very small bars, cast under the most favorable conditions, precisely as similar and equally high results have been obtained on the old gun-bronze. My experience with it in the testing-machine is, that it is a material about one-quarter stronger than the old gun-bronze, and I should be much surprised to see it exhibit a greater tensile strength than 30,000 pounds on a bar 2 inches in diameter and 15 inches between marks. So far as non-corrodibility and casting-qualities are concerned, manganese-bronze is all that gun-bronze was. It is extensively and successfully used.

Aluminum-bronze is by long odds the strongest of all the copper-alloys. A small bar of it, cast in sand, broke in the testing-machine of the Leeds Forge Company, at Leeds, England, in the presence of Mr. Sampson Fox, the inventor of the corrugated boiler-tube, at the unheard-of load, for a copper-alloy, of 130,000 pounds per square inch, and with about one-half of one per cent. elongation in one inch. The bronze in question was made from corundum at Lockport, N. Y., in the Cowles electric furnace, and had approximately 9½ per cent. of aluminum, 1⅛ of silicon, 1½ of iron and 88½ of copper in its composition. It is what is known as the "A 1" grade of bronze. Many hundreds of tests on this grade of metal, on small bars up to three-fourths of an inch in diameter, prove its tensile strength to be over 100,000 pounds on bars of that size, while its transverse strength, on one test, ran as high as 9500 pounds on a bar 1 inch by 1 inch by 12 inches. Its elastic limit is proportionately high, and its hardness far above all other alloys. Its corrodibility is much less in salt water than that of either manganese- or gun-bronze.

To those designers who desire a material that will stay in place when the engines are exerting their greatest power, the stiffness and high elastic limit of A 1 bronze render it an ideal metal. That it is a short metal, so far as ductility is concerned, should not weigh against it, for it is in no sense a brittle metal. Even the hardest

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and strongest pieces of it will bend a trifle, and its resistance to shock is remarkable. But while these are its advantages, it is, like all good things, not without its imperfections. It must be properly cast, by those who are familiar with it, to obtain good results. To founders who do not understand it, no more refractory metal can be given. Still, there is nothing in the handling of this material more difficult to overcome than what has been done in countless instances in the working of steel. Its price alone is the one great obstacle to its introduction into propellers generally. This is, approximately, $700 per ton for the ingot-metal alone.

Cowles aluminum-brass will be found to be a material that has the maximum of good qualities sought for by the propeller-makers at the minimum cost. It is a patented composition. This alloy is made by fusing together equal weights of A 1 aluminum-bronze (that made from corundum is the best for this purpose, the silicon and iron being necessary elements, if it is desired to obtain the highest possible strength), copper, and zinc. The copper and bronze are first thoroughly melted and mixed, and the zinc is finally added. The material is left in the furnace until small test-bars are taken from it and broken. When these bars show a tensile strength of 80,000 pounds or over, with 2 or 3 per cent. ductility, the metal is ready to be poured. Tests of this brass, on small bars, have at times shown as high as 100,000 pounds tensile strength. Three of these bars, which were attached to a large casting made by the Brush Electric Company, at Cleveland, for the hub of the "Colossus" dynamo in the Cowles works at Lockport, exhibited a tensile strength of from 93,000 to 96,000 pounds, and 2½ to 7 per cent. elongation. A bar of it 2 inches in diameter and 15 inches long was broken at 82,000 pounds on the Watertown machine by Chief Engineer Harris, U. S. N., whose results appear in a letter appended to this paper, along with other tests of these alloys. Three tests of this No. 2 brass in England, on sand-castings, gave equally high results. A bar 1 inch by 6 inches broke at 76,187 pounds, with an elongation of 1½ per cent. Another bar, 2 inches by 12 inches, broke with a load of 72,912 pounds per square inch and 1 per cent. elongation. The last bar was 3 inches in diameter and 18 inches long between marks; it broke at 66,752 pounds and a slight elongation. These three tests were made at the proving-house of Lloyd's, near Netherton, and, in conjunction with the American tests, are most conclusive. The brass was made at the Cowles electric smelting-plant at Milton, near Stoke-on-Trent. The 3-inch bar above referred to was cast attached to a large propeller

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blade. Its elastic limit is undoubtedly almost as high as that of A 1 aluminum-bronze, and its real strength must certainly be fully twice or three times that of manganese-bronze, while its casting qualities are all that can be desired. The screw of the United States gunboat Petrel is cast from this brass, mixed with a trifle less zinc, in order to increase its ductility. Its cost is about the same as that of other alloys. Our No. 1 aluminum-brass has less tensile strength, but 10 to 15 per cent. ductility.

In reviewing the various alloys and metals used in screws, I have in but one instance mentioned the price of the material, for the reason that the cost of the iron and steel wheels are patent to us all, and that, with but one exception, the prices of the alloys are so nearly the same that quality should alone be considered in selecting the proper one for this purpose.

In passing upon the various materials above cited, a rather confused idea of the strength of the several alloys may appear to prevail. This is due to the fact that we have not to-day, in this country or in England, any official standard by which to judge of the physical characteristics of cast metals. The Harris test-bar is 22 inches long, 2½ inches in diameter at the head, and 2 inches in diameter in the shank, and 15 inches between elongation marks. Cowles aluminum-bronze and brass, the United States Government gun-bronze, the "de-oxidized" bronze from Bridgeport, and Cowles manganese-bronze have thus far been tested at Watertown on bars of the above dimensions. At the first glance this would seem to be all that could be desired; but there are two other conditions that are absolutely necessary to be known before we can make a fair comparison of different materials: namely, whether the casting was made in dry or green sand or in a chill, and whether it was attached to a larger casting or cast by itself. It has been the experience at the Cowles Lockport works, where all of the non-ferrous alloys have been tested at great expense and labor, that, as a rule, the smaller the bar the higher the result, no matter what alloy is under consideration, unless, perhaps, it be the aluminum-brass. Tin-bronze and manganese-bronze show this property as much as aluminum-bronze; it is also true of the ordinary brasses. It has also been found that chill-castings give higher results than sand-castings, and that bars cast by themselves purposely for testing almost invariably run higher than test-bars attached to castings. It is also a fact that bars cut out from castings are generally weaker than bars cast alone. From the above it will be seen that if a fair comparison is to be had, it is of the utmost im-

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portance that all test-bars of cast metals must be made under identically the same conditions in the foundry. Where the material is supposed to be the same, or approximately so, a complete analysis of the broken bars should be made.

To sum up the advantages claimed for aluminum-brass:

Its enormous transverse strength permits far thinner blades being made of it than can be made of any other alloy.

Its greater hardness permits it to take a sharper edge, and to hold it longer than other alloys. It is now the practice of the best constructors to sharpen the edges of their wheels until they are almost like a knife.

Its unequalled elastic limit keeps the wheel in perfect form, and true to its mathematical design, under far heavier strains than can be withstood by any metal except steel.

Its higher resistance to corrosion preserves the brightness and smoothness of the face of the blades, and thus renders friction less than in other alloys.

Its greater freedom from galvanic action permits the use of less zinc plate, and makes it less expensive to operate than any other alloy. This cost of zinc plates is an item of expense that amounts to quite an annual percentage on the cost of any bronze propeller.

In short, the lightest, strongest and most non-corrodible propeller yet known may be made from aluminum-bronze or brass, and the 25- or 30-knot vessels of the near future will only be possible by reason of its use.

Report of W. H. Harris, Chief Engineer U. S. N.

U. S. Navy Department, Bureau of Steam Engineering,
Washington, December 20, 1887.

Sir: In obedience to orders of the 22d ultimo, I have visited Watertown, Mass., and Lockport, N. Y., to witness experiments at the former place upon specimens of aluminum-bronze and brass, submitted for tensile tests by the Cowles Electric Smelting and Aluminum Company, of Lockport, N. Y., which place I visited after the tests at Watertown were completed.

In competition with the bronze specimens furnished by the above company, the bureau directed that six specimens of the tin-bronze, usually used for propeller castings in the United States naval service, be tested under like conditions.

All specimens tested were 15 inches in length between reference marks, and, if finished all over, were 1⅞ inches in diameter. Those which were left as taken from the sand, were slightly in excess of this diameter, 1.93 inches being the greatest diameter recorded.

A tabulated statement of the results of the tests is appended to this report.

All of the above aluminum-bronzes can be worked hot or cold, but no ex-

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periments have been made, to my knowledge, to determine their strengths after such working.

It makes sound and sharp castings. Its greater tensile strength allows the use of thinner and consequently lighter blades for propellers, and its great ductility allows of its being bent to nearly a right angle without showing cracks or flaws. I have no knowledge of any alloy of copper which, as a casting, combines the qualities which this material possesses.

Very respectfully,

WILLIAM H. HARRIS,
Chief Engineer U. S. N.

The following tables and remarks cover the details of the tests made by Engineer Harris, and are from the *Proceedings* of the United States Naval Institute:

Harris Tests of Aluminum-Alloys.

Marks	1 C	7 C	9 C	10 C	11 C
Length, inches	15	15	15	15	15
Diameter, inches	1.87	1.87	1.87	1.87	1.87
Area, inches	2.76	2.76	2.76	2.76	2.76
Diameter at fracture, inches	1.56	1.8	1.66	1.82	1.78
Tensile strength per sq. in., lbs.	60,700	66,000	67,600	72.830	82.200
Elastic limit, " " "	18,000	27,000	24,000	33,000	60 to 73,000
Elongation, per cent	23.2	3.8	13	2.40	2.33
Reduction of area, per cent	30.7	7.8	21.62	5.78	9.88

Harris Tests of Aluminum-Alloys.

Marks	11 D	13 C	9 D	10 D	13 D
Length, inches	15	15	15	15	15
Diameter, inches	1.9	1.87	1.9	1.87	1.87
Area, inches	2.84	2.76	2.84	2.76	2.76
Diameter at fracture, inches	1.86	1.64	1.75	1.86	1.73
Tensile strength per sq. in., lbs.	70,400	59,100	53,000	69,930	46,550
Elastic limit, " " "	55,000	19,000	19,000	33,000	17,000
Elongation, per cent	0.4	15.1	6.2	1.33	7.8
Reduction of area, per cent	4.33	23.59	15.5	3.30	19.19

Harris Tests of Navy Gun-Bronze.

Marks	1	2	3	4	5
Length, inches	15	15	15	15	15
Diameter, inches	1.87	1.87	1.87	1.88	1.88
Area, inches	2.76	2.76	2.76	2.77	2.77
Diameter at fracture, inches	1.83	1.81	1.77	1.78	1.82
Tensile strength, per sq. in., lbs.	18,000	24,590	22,500	23,000	20,570
Elastic limit, " " "	10,000	11,000	11,000	13,000	13,000
Elongation, per cent	2.5	8.2	5.8	4	2.33
Reduction of area, per cent	4.7	6.8	10.89	10.36	6.3

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Approximate Composition of the Above Alloys.

Mark or Number.	Copper. Per cent.	Aluminum. Per cent.	Silicon. Per cent.	Zinc. Per cent.	Iron. Per cent.
1 C	91.50	6.50	1.75	.....	0.25
7 C	88.50	9.33	1.66	.....	0.50
9 C	91.50	6.50	1.75	.....	0.25
10 C	90.00	9.00	1.00	.....	.....
11 C	63.00	3.33	0.33	33.33	.....
11 D	63.00	3.33	0.33	33.33	.....
13 C	91.50	6.50	1.75	.....	0.25
9 D	93.00	6.50	0.50	.....	.....
10 D	88.50	9.33	1.66	.....	0.50
13 D	92.00	6.50	0.50	.....	.....

The approximate composition of all the five samples of gun-bronze was: copper, 88 per cent.; tin, 10 per cent.; and zinc, 2 per cent.

The following table shows the results of tests on the hardness of the large bars:

Hardness of Aluminum-Bronze or Brass.

Mark.	Head.	Stem.
1 C,	9.39	13.85
7 C,	14.12	14.26
9 C,	11.18	13.59
11 C,	14.69	10.60
10 D,	17.08	16.07
Navy Gun-Bronze.		
No. 1,	3.33	3.33
No. 5,	3.67	6.56

The bars furnished by the government, and marked "navy bronze," were carefully cast at the New York Navy Yard, under the inspection of an officer detailed for that purpose, and were exceedingly fine castings. You will notice that they are of the composition of ordinary gun-bronze.

Two more bars of Cowles aluminum-bronze, cast in green sand, of the same grade as above, were officially tested for Lieutenant Martin E. Hall, U. S. N., at the Washington Navy Yard, with the following results:

Grade of Specimen A 1.	Dimensions of Specimens.		Weight applied.	Elastic Limit per Sq. In. on Original Cross-Section.	Weight per Sq. In. on Original Cross-Section.	Elongation.	Reduction of Area.
	Length.	Diameter.					
	in.	in.	lbs.	lbs.	lbs.	P. c.	P. c.
No. 1.....	2.0	.500	22,483	....	114,514	0 45	0.00
No. 3.....	2.0	.501	21,650	79,894	109,823	0.05	2.39

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PHYSICAL PROPERTIES OF SOME OF THE ALLOYS OF MANGANESE, COPPER AND ALUMINUM.

BY EUGENE H. COWLES, LOCKPORT, N. Y.

(Washington Meeting, February, 1890.)

THE German silver industry of the United States amounts in value to upwards of $6,000,000 or $8,000,000 annually. Several thousands of people earn a livelihood pursuing it, and the beautiful goods and articles manufactured from this time-honored and valued alloy are everywhere visible.

German silver enters largely into our street signs, harness and house hardware, and all manner of table-ware, solid or plated. There is scarcely an hour in the day that German or nickel-silver is not before our eyes.

The composition of this alloy is a very uncertain thing, and depends largely on the honesty of the manufacturer and the price the purchaser is willing to pay. It is composed of copper, zinc and nickel in varying proportions. The best varieties contain from 18 to 25 per cent. of nickel, and from 20 to 30 per cent. of zinc, the remainder being copper. The more expensive nickel-silver contains from 25 to 33 per cent. of nickel, and from 75 to 66 per cent. of copper. The nickel is used as a whitening element; it also strengthens the alloy and renders it harder and more non-corrodible than the brass made without it, of copper and zinc. Moreover, it reduces the odor emanating from brass, which is so disagreeable if it occurs about the dining-table.

Of all troublesome alloys to handle in the foundry or rolling-mill, German silver is the worst. It is unmanageable and refractory at every step in its transition from the crude elements into rods, sheets or wire.

As a white-headed furnace-man recently said to the writer, in one of our large mills, "you can put eight crucibles into the fire, each containing exactly the same mixture of German silver, and when you take them out and pour them into slabs for rolling, seven will be a mass of scum or slag, or the stuff will swell up like rising bread and overflow the moulds, and be so full of blow-holes that it will look like a sponge. The eighth crucible will contain perfectly good metal. On remelting the next day, the bad stuff will, in nine chances out of ten, come out just as good metal as the eighth pot the day before, and under apparently the same conditions."

With nickel at seventy or eighty cents per pound, German silver

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is necessarily an expensive metal, even in the ingot, and its extremely refractory nature makes it a still more costly luxury when fabricated into goods. With a view to obtain, if possible, a cheaper and better article than German silver, my brother, Alfred H. Cowles, and myself began some years ago to experiment with the alloys of copper and manganese. We found that while a pure metallic manganese could with difficulty be reduced by the ordinary methods, it could be cheaply reduced in the electric furnace. After a long course of experiments we have perfected a substitute for German silver, which is offered to the public by the Cowles Electric Smelting and Aluminum Co., under the name of "Silver-Bronze."

The genesis of this alloy is somewhat interesting. Of course, it was no new thing to bleach copper with manganese. Dr. Percy, the discoverer of aluminum-bronze, was also the discoverer or inventor of manganese-bronze. To him, and to him alone, is due the honor of having first made these two alloys.

Following his written description, several other makers have from time to time, within the past thirty years, attempted to produce a pure manganese-bronze, but without success. Some have even attempted its introduction into the German silver trade. But thus far it has been both too expensive to manufacture and too corrodible to permit a prolonged use.

In order to overcome the two prime difficulties, that of casting and that of corrosion, we have introduced a small percentage of aluminum into the alloy, with the happiest effect.

The successive steps that have been pursued are recorded in a list of upwards of two hundred distinct mixtures of the several metals, copper, zinc, tin, lead, aluminum, iron and manganese and the metalloid silicon, and experiments upon the same in ascertaining tensile strength, ductility, color, etc. The most important determinations derived from this list of experiments appear to be about as follows:

1. That pure metallic manganese exerts a bleaching effect upon copper more radical in its action even than nickel. In other words, we found that 18½ per cent. of manganese present in copper produces as white a color in the resulting alloy as 25 per cent. of nickel would do, this being the amount of each required to remove the last trace of red.

2. That upwards of 20 or 25 per cent. of manganese may be added to copper without reducing its ductility, although doubling its tensile strength and changing its color.

3. That manganese, copper and zinc, when melted together and

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poured into moulds, behave very much like the most "yeasty" German silver, producing an ingot which is a mass of blow-holes, and which swells up above the mould before cooling.

4. That the alloy of manganese and copper by itself is very easily oxidized.

5. That the addition of 1.25 per cent. of aluminum to a manganese-copper-alloy converts it from one of the most refractory of metals in the casting process into a metal of superior casting qualities, and the non-corrodibility of which must in many instances be far greater than that of either German or nickel silver.

The "silver-bronze" alloy especially designed for rods, sheets and wire has the following composition: Manganese, 18; aluminum, 1.20; silicon, 5; zinc, 13, and copper, 67.5 per cent. It has a tensile strength of about 57,000 pounds on small bars, and 20 per cent. elongation. It has been rolled into thin plate and drawn into wire .008 inch in diameter. A test of the electrical conductivity of this wire (of size No. 32) shows its resistance to be 41.44 times that of pure copper. This is far lower conductivity than that of German silver, and is a matter of considerable importance to electricians who may wish a cheaper and better material than can be had at present, from which to construct resistance-coils.

Postscript.

The following letter, received since the foregoing paper was issued in pamphlet form, contains the final determination of the electrical characters of the silver-bronze alloy:

Case School of Applied Science,
Cleveland, Ohio, April 22, 1890.

I have determined the resistance and change of resistance of The Cowles Silver-Bronze, as follows:

The resistance of one meter of the wire, one mm. in diameter, is 0.996 ohms.

The change of resistance per degree centigrade is less than 0.00034 per cent. Its resistance is therefore about 52 times that of silver, about 48 times that of copper, and about 3.7 times that of German silver, and hence considerably greater than that of any other material known which is capable of being drawn into a strong, tough wire.

Its change of resistance for temperature is only about one-hundredth, *at most*, of that of German silver. My apparatus was only delicate enough to determine this as a superior limit. This makes its change of resistance for temperature far less than that of any known substance.

Harry Fielding Reid, Ph.D.,
Professor of Physics, C. S. A. S.

This metal is now known at the Brush works as "the metallic insulator," and from our electrical standpoint is really a wonderful material.

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WURTZILITE FROM THE UINTAH MOUNTAINS, UTAH.

BY WILLIAM P. BLAKE, NEW HAVEN, CONN.

(Washington Meeting, February, 1890.)

In addition to uintaite, the Uintah Mountains contain a deposit, or vein, of the peculiar hydrocarbon mineral, to which I have given the name wurtzilite,* in honor of the chemist, Henry Wurtz, Ph.D., of New York, who, in 1865, described grahamite, and who has added to our knowledge of the composition of the hydrocarbon minerals.

Wurtzilite and uintaite are very similar in appearance, but are very different in their physical and chemical properties. Both are black, splendent and shining in fresh fracture; but the wurtzilite has a decided toughness, and a degree of elasticity, especially when warmed, which suggests to some persons a resemblance to caoutchouc. It was at first, at Salt Lake, believed and reported to be a mineral form of India-rubber, and great expectations of its value as a substitute for crude rubber were indulged in. It was, however, found that it would not act like rubber with sulphur; it would not incorporate with it and become "vulcanized," and it was very refractory and insoluble in the ordinary solvents of rubber and of uintaite.

The elasticity, though unlike the elasticity of rubber, led to the reference of the mineral to the species *elaterite*, the only name under which an elastic bitumen is described in the mineralogies; and it is so called in the list of the mineral products of Utah, given in the last message of the Governor of the Territory.

The name elaterite, as used by Dana, covers at least three differing substances, ranging in specific gravity from 0.905 to 1.223. Two of the varieties are soft and elastic, much like India-rubber, and one is occasionally hard and brittle, imbedded in the softer kinds. All are rare, and comparatively unknown, and not generally represented in mineral collections. One of the varieties referred to here is the *subterranean fungus* from Derbyshire, described by Lister as early as 1673.

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* *Engineering and Mining Journal*, xlviii., December 21, 1889, p. 542..

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Since the publication of my notice, two important communications on the subject have appeared from Dr. Wurtz, who has studied the solubility, fusibility and other peculiarities of wurtzilite.*

A comparison of wurtzilite with the varieties of elaterite, as described, shows that it is radically different from either of them, and that it has much better defined characters, which permit of a specific description, and entitle it to a distinctive name. These characters are as follows:

Form.—Amorphous; in large, homogeneous masses, free from mechanical mixture or contamination with earthy substances. It is without any evidence of formation in layers, or of columnar structure, but its condition in the bed, or vein, is not yet known or not described. When cold, or at ordinary temperatures, it is brittle, and breaks with a large conchoidal fracture, giving very brilliant, glossy surfaces, with a vitreous lustre, like uintaite, and resembling in appearance the splendent conchoidal surfaces of newly-broken obsidian. It has, however, a decidedly tough quality, which increases with an elevation of temperature, and it requires a quick, sharp blow to detach a flake and secure a good fracture. It shows a tendency to break in thin flakes, rather than in cuboidal or spheroidal masses. In the midst of a large mass of the mineral, remarkable for the breadth of the conchoidal surfaces, there is a small nodular mass, with a granular fracture. The color and lustre are, however, the same as in other portions, and there does not appear to be any essential difference in composition; though, as yet, no quantitative determinations have been made.

Elasticity.—In thin flakes it is decidedly elastic, and when bent, flies back like a spring. This elasticity may best be compared to that of glass, or mica, rather than to the yielding or tensile elasticity of India-rubber. It is not like a mass of rubber, not being impressible by the nail, or a hard substance, and it cannot be drawn out like rubber. It may be said to be flexible and sub-elastic. If a thin flake is bent too quickly, or too far, it snaps like glass; but if warmed, a flake may be bent nearly double. It is sectile, difficult to reduce to powder, and cuts a little like horn or whalebone.

The nearest approach to the peculiar toughness and elasticity of wurtzilite among the bituminous coals is found in the Breckinridge cannel coal of Kentucky. The wood-like resilience characteristic

* *Engineering and Mining Journal*, xlix., January 11, 1890, p. 59, and January 25, p. 106.

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of this coal was first noticed and described by Prof. Silliman, in 1854.*

Hardness.—It scratches gypsum and does not scratch rock salt. Its hardness is thus between 2 and 3 on the scale of Mohs, but it is nearer 2 than 3.

Color.—The extremely thin edges of the flakes obtained by fracture are garnet red by transmitted light, often brilliantly so. It would thus appear to be transparent in very thin plates, and to be deep red in color, though the color of the mass, by reflected light, is a glossy jet black, like uintaite.

Specific Gravity.—Two trials on different fragments in pure water at 60° F. gave respectively 1.0238 and 1.0306, the mean of which is 1.0272, approximating 1.03. Dr. Wurtz has since taken the gravity both in water and in alcohol with extreme care, and obtained from five different samples the following: 1.0265 in water; 1.022 in alcohol; 1.0236, 1.0226, 1.0227 in alcohol. Of these figures he prefers the fifth, 1.0227, which is also the mean of Nos. 2, 3, 4 and 5, rejecting the first as too far from the mean.

Fusibility.—It does not fuse or melt in boiling water, but becomes softer, tougher and plastic and more elastic, but is not viscid or sticky, and fragments do not cohere.

Brought near to the flame of a candle, it softens and melts, takes fire, and burns with a bright luminous flame with a slight crepitation and little smoke, giving off gas and a strong bituminous odor. Heated in a glass tube, it gives off a dense cloud of white and yellow smoke, and distils over a thick brown tar-like oil with a strong odor, and leaves a very small residue of fixed carbon.

According to the view of Dr. Wurtz,† the mineral, unless under pressure, begins to decompose below its melting-point, and it does not fuse below 800° F. It is not fused in boiling paraffine, and undergoes little or no change in boiling sulphur, according to Dr. Wurtz.

Solubility.—It is not easily dissolved, and resists the usual solvents of bitumen, but its deportment with solvents and other chemical characteristics have not been studied by me, but at my request are under investigation by Dr. Wurtz. He has found that alcohol does not dissolve wurtzilite, while, according to Johnston, it has considerable solvent action upon elaterite. He also finds that prolonged

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* *Proceedings Am. Assoc. for the Advancement of Science*, May, 1854.

† *Engineering and Mining Journal*, xlix., January 11, 1890, p. 59.

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digestion in pure ethyle oxide extracts 4.17 per cent. by weight of the mineral. This extract is a yellow oil with a disagreeable odor, suggestive of stramonium. It has remained liquid for a month or more without any apparent change, except a partial loss of odor. Extracts by ether from this class of substances are generally solids or semi-solids. The removal of this oily substance from wurtzilite does not appreciably affect the elasticity, thus showing that this peculiar characteristic is not dependent upon the presence of the oil. Hot nitric acid effects a partial solution, leaving a soft, translucent, brown magma, which does not appear to be a nitrated product. "The brown nitric acid solution becomes turbid on cooling, and water precipitates further a considerable amount of a light-brown flocky material."*

The deportment of this mineral with boiling paraffine is in strong contrast with that of uintaite, which is readily dissolved.

Electrical Characters.—It is a good electric; negative electricity is easily developed in it by friction. It will be found valuable as an insulating material, and for other purposes in the arts. Mr. Gilson, of Salt Lake, has already experimented with the mineral, and he gave me a sample of copper wire thickly coated with a flexible insulating covering, the basis of which is wurtzilite.

Comparison of Wurtzilite and Elaterite.

Dr. Wurtz, in his communication, January 11th, has set forth so carefully and fully the characters of elaterite as described by Hausmann, Lister, Hatchett and others, and has also so compared them with wurtzilite, that for convenience of reference I quote from his paper. The principal results stated in his communication of January 25th have already been briefly given under the appropriate headings.

The principal question is in regard to the placing of this substance under the head of Hausmann's so-called *elaterite*—a species adopted, with this name, by such eminent authorities as Leonhard, Beudant, Haidinger, and last, but by no means least, Dana. The name itself and its etymology, when examined, seem to involve a sort of jest, and it is hard to assign to them any scientific significance. The elaterium or "squirting cucumber," however, suggests, if anything, what Prof. Blake well calls "tensile elasticity," as distinguished from mere elasticity of flexure, a matter of import here.

The kind of elasticity referred to by the most ancient authority on "elaterite,"

* Wurtz, "Some Further Observations on Wurtzilite," *Engineering and Mining Journal*, xlix., January 25, 1890, p. 107.

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Lister, in 1674, is that of caoutchouc, or what we might designate as *gelatinous* elasticity. Lister says of the Derbyshire mineral that it is "much like peats or turff cut up in the high moors, both in the sooty color and inward substance; this only is more clammy and tough, and dries not. And some of this fungous substance is very soft and like gelly." Hatchett, in 1797, wholly unaware of Lister's account of more than a century previous, and attributing the first discovery of the Castleton (Derbyshire) mineral to a Mr. de Born in 1787, describes characteristic varieties as follows:

—— "Dark yellowish-brown, elastic, very soft and adherent to fingers.

—— "Brownish-olive, becoming reddish-brown in the air. Opposed to the light it appears semi-transparent, and of a yellowish-brown inclining to orange; soft, elastic and, when recently cut, adherent to fingers.

—— "Same as the last; darker brown and harder. *Specific gravity* = .9053.

—— "Dark brown; harder than the last. *This exactly resembles cahoutchou* in the *degree of elasticity*,* and in the property which it possesses of removing the traces of black lead."

An appropriate interpolation here will be the remark that Prof. Blake's wurtzilite marks ordinary foolscap paper with a distinct brown streak, and writes its own name quite legibly. Hatchett introduces his subject with the remark that this "new species of bitumen much resembles in elasticity and color the substance known by the name of cahout-chou, or Indian rubber." It is proper to add that he describes one variety as follows: "Reddish-brown, perfectly hard and brittle. The characters of asphaltum are complete in this specimen. Specific gravity = 10.233" (doubtless a misprint for 1.0223). This variety is the only one Hatchett describes that approximates in any respect to wurtzilite, but Hatchett compares it to asphaltum, with which no mineralogist would place Prof. Blake's mineral. Its characteristic sectility and cohesion, with its infusibility, etc., would forbid this. Hatchett says that in his Castleton specimens all the grades of modification between petroleum and asphaltum could be traced, "the elasticity being most complete in the variety which occupies the middle place between petroleum and asphaltum."

One of his varieties, moreover, *resembles cork in color and texture when recently cut.* This forms exterior coatings on the more compact varieties described above. The specific gravity of this was .9881 and .9748, mean = .9815. This variety passes further into an "ochraceous-colored powder." The second variety mentioned above, on which Hatchett lays stress, as characteristic of the species, is partly dissolved by ether at 55 degrees (Fahrenheit, doubtless). The solution is yellowish-brown by transmitted and brownish-olive by reflected light. This would import the presence of some fluorescent matter, similar to that in many crude petroleums. Evaporation of the ethereal solution gives a "yellowish-brown bitumen," constituting 18 per cent. of the original, "totally devoid of elasticity." The cork-like variety is little affected by ether.

I have, so far, been able only to make some preliminary tests with ether on shavings of wurtzilite, too incomplete to carry much weight. The ether becomes colored pale yellow, taking up a small portion of the mineral not yet examined. Klaproth (1802) quotes Hatchett, and states that he has himself examined several varieties of the mineral, which he calls "Elastisches Erdpech," in compact pure fragments of brownish-olive color, which, against the light, is semi-transparent, and transmits a bright hyacinth-red tint. It is *soft, very elastic and adheres to the fingers.* Insoluble in all liquids tried by him (he does not mention ether). Absorbs petroleum and

* The italics are mine.—H. W.

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swells up and becomes softer. Boiling fuming nitric acid and strong alkaline lyes are without appreciable action on it. Fuses at a high heat, and after this may be drawn into threads (zu Fäden ziehen) between the fingers. After fusion, soluble in petroleum and other oils. No specific gravity stated. It contained 6 to 7 per cent. of reddish-brown ash, alkaline in reaction.

The first edition of Hausmann's *Handbuch* (1813) I have not access to. In the edition of 1847 he describes "elaterite," among other characters, as "very soft," at times somewhat pasty; as elastic as Kahutschuk; odor bituminous; easily fusible." Very little soluble in alcohol; occurring in kidney-shaped or mushroom-shaped masses. He cites the analyses of Henry, Jr. (1826), and of Johnston (1838), and concludes that it probably has the composition CH_2 corresponding to ozocerite.

The most important record regarding the assumed species, elaterite, is doubtless the paper of Prof. J. F. W. Johnston, of Durham, which I have succeeded in finding in the *Lond. and Edinb. Phil. Mag.*, vol. xiii., 1838, p. 22. His analytic computations require revision, apparently, as I have already indicated in your *Journal* more than ten years ago, with regard* to analyses in general of carbo-hydrogen minerals made previous to 1841, when the true equivalent of carbon was first made known by Dumas and Stas. The discussion of these analyses, therefore, would require more time than I now have, and probably more space than you would give, and must be deferred.

Prof. Johnston's description of what he calls "Elastic Bitumen of Derbyshire" covers three varieties: 1. Soft, elastic, adheres to fingers, yields to slight pressure, brown, and of peculiar odor. Lost weight and an odorous volatile matter at 212 degrees F. 2. Closely resembling moderately soft india rubber; dark brown. When boiled in water, "a volatile portion collected on the surface of the water and the sides of the flask, which, on cooling, presented the appearance of a very soft white or slightly brownish solid. Boiling alcohol and ether extracted a similar volatile substance, but very sparingly."

Wurtzilite—at least as represented by Prof. Blake's specimens—differs from Hausmann's so-called *elaterite* in the following particulars:

"Specific gravity, which Hatchett makes, in his characteristic *elastic* varieties, less than that of water, namely, as low as .9053. Prof. Blake's figure for wurtzilite is 1.03. I obtained, in water, from some very pure and brilliant flakes selected for me by Prof. Blake, 1.026; and for some larger pieces selected by myself, taken in alcohol, 1.022. I propose making further determinations of this figure, but the evidence is sufficient that the mineral is heavier than water.

"The *elasticity* of wurtzilite, on which so much stress has been put, is, in the characteristic variety (if varieties there are) to which the name has been given, as it appears to me, no characteristic at all. Probably all solids possessing rigidity and cohesion show in thin sheets or flakes this kind of elasticity of flexure. Cannel coal is a familiar example among carbo-hydrogen minerals. The elaterite elasticity is clearly of the *gelatinous* kind.

"A true characteristic of wurtzilite is the combination of the splendent vitreous fracture with the remarkable degrees of sectility and hardness. This alone separates it from all the varieties of *elaterite*, of which I have so far encountered descriptions.

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* *Engineering and Mining Journal*, April 5, 1879, in an article on the "Utah Ceroid Minerals." See foot-note, in which allusion is made to the fact that Prof. Dana has overlooked this important matter *in some* of his citations. The many analyses of Johnston are, at least in part, included in this remark.

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"Its behavior with both alcohol and ether distinguishes wurtzilite from *elaterite*. On the latter, alcohol is stated by Johnston to have considerable solvent action. My own experiments show me that long soaking in anhydrous alcohol does not even affect the brilliant natural lustre of the fractures of wurtzilite.

"Hausmann makes his *elaterite* 'easily fusible,' but there is a lack of consentaneity among the authorities on this point, going to prove, together with many other circumstances, that this name has been used to cover several distinct materials. Wurtzilite, unless under pressure, begins to decompose below its point of fusion.

"Several of the points of chemical character and behavior mentioned by Prof. Johnston are not recognizable in the case of the wurtzilite. Johnston's later paper on the 'Settling Stones Resinoid' (quoted by Dana under 'Elaterite') is still wider of the mark."

My general conclusion must be that Prof. Blake's mineral is certainly not to be rationally put under Dana's heading of elaterite, and is a well-characterized new species. I shall entertain the hope of getting some analyses of the mineral, both proximate and ultimate, in the course of a short time.

NOTE ON THE USE OF ALUMINUM IN THE CONSTRUCTION OF INSTRUMENTS OF PRECISION.

BY WILLIAM P. BLAKE, NEW HAVEN, CONN.

(Washington Meeting, February, 1890.)

THE adaptation of aluminum to the construction of portable instruments of precision, where lightness is important, is well illustrated by the double reflecting and repeating circle, the invention of Captain Charles Hervey Townshend, of New Haven.

One of these instruments, now exhibited to the members, has just been completed by Stackpole & Bro., of New York, and is made of aluminum, with the exception of a few minor pieces and a portion of the handle. This instrument is intended for use in a boat, and to be held in one hand like a sextant; and as it is a complete circle, lightness of the material becomes more important.

The whole instrument, as shown, exclusive of the eye-pieces and the handle, weighs only one pound, or about one-third of the weight of a sextant of the usual construction. The circle, about nine inches in diameter, was cast in one piece from metal furnished by the Pittsburgh Company.

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The mere inspection of this circle, and of the limbs of the instrument, is sufficient to show that the metal works well under the file, in the lathe, and under the graduating tool. The castings are homogeneous, free from blow-holes, and dress up clean and sharp. Every part, also, appears to have the requisite rigidity under the touch, although the extreme lightness raises a doubt of it in the mind.

There is also before the Institute another example of the use of aluminum in the construction of instruments; a mining transit, shown in the exhibition of fine instruments by Fauth & Co., of Washington.

The great advantage of lightness in such instruments will be appreciated by those who have had to carry instruments of the usual construction through the devious passages of mines, and up the sides of high mountains, where every ounce of weight is a grievous burden.

Another great advantage which aluminum has over brass is its resistance to corrosion by the atmosphere, moisture and even sea-water. It does not require a coating of lacquer, as all brass instruments do, and its unprotected surface keeps clean and bright where brass and ordinary bronze metal will corrode and become green and dirty.

But with these great advantages offered by the commercial aluminum, it is probable that it can be improved for general instrument work by the addition of a small amount of silver. The aluminum, it must be said, is rather soft for wearing parts, such as bearings and rubbing-surfaces, and it is perhaps not quite as rigid and stiff as it should be for some parts of instruments. The alloy shown here by Mr. Hunt, consisting of 95 parts of aluminum by weight and 5 parts of silver, while but a little heavier than the aluminum, is much more rigid and harder, and works quite as well, or better, under tools. The specific gravity is about 3.2, aluminum being 2.6. This alloy is whiter than the aluminum, more like silver, and withstands the corrosive action of the atmosphere and sulphur nearly as well. It takes a good polish, and works well under the graver. It is better for graduation.

The leading instrument-makers are yet very cautious in the use of the unalloyed aluminum. Messrs. Fauth & Co., of Washington, make considerable use of aluminum in astronomical apparatus, where great rigidity is not required and lightness is important, and which have careful handling. They find, in working the metal, that it is apt to tear under the tool, and that it does not give good

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clean threads. Others, however, who have worked with the metal, have overcome this last difficulty by a little care in the use of the tap and die.

The Keuffel & Esser Co., of New York, have been experimenting with aluminum in their shops, and have made some sextants of it which have proved to be satisfactory as far as the limited experience in the use of them has shown. They have also a mining transit of aluminum in construction, some portions of the instrument being, however, made of a harder and heavier metal.

While the unalloyed aluminum is no doubt soft and somewhat "spongy" in working, the evidence presented in the finished instrument exhibited at this meeting is sufficient to show that the metal and its alloys have great merit as materials for the construction of instruments of precision.

THE RUSH CREEK, ARKANSAS, ZINC-DISTRICT.

BY H. M. CHANCE, PHILADELPHIA, PA.

(Washington Meeting, February, 1890.)

ONE hundred and twenty miles of hard travel over steep ridges almost due east from Fayetteville, on the St. Louis and San Francisco Railroad, about 100 miles from Eureka Hot Springs, or 75 miles northwest from Batesville, will bring the traveller to the waters of the Buffalo Fork of the White river, in a wild, thinly-settled, heavily-timbered country, now being prospected quite actively for zinc, and known as the Buffalo or Rush Creek district. Excepting here and there a piece of bottom land, patented before the discovery of zinc, the land is unoccupied Government land, and subject to entry under the United States mineral laws.

Claims are now being taken, 600 by 1500 feet, but a movement made this summer, if successful, will largely modify the position of claim-holders. A few claims are now being filed under the Placer Act on the ground that as the zinc occurs as a horizontal, sedimentary deposit, etc., it can only be properly so classified.

The zinc occurs in limestone, which is at times so siliceous that

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it might well be called quartzite. The northern tier of counties in Arkansas is almost entirely underlaid with limestones, which, beginning at the top with the Subcarboniferous, immediately beneath the coal-measures, extend down in the geological scale an unknown distance—possibly to the lower Silurian—an unbroken succession of limestones, with intercalated beds of chert or flint, of great thickness. The zinc-horizon is probably low down in this great mass of limestones, possibly near the bottom of the series.

In what is known as the Rush Creek district, at least two mineralized zones exist, one in the hills, 100 to 150 feet above water-level, and the other some 70 or 80 feet below water-level. The first has been located on a large number of claims, but on only three or four has sufficient work been done to show much mineral in place. Other zones will doubtless be found when some systematic development work is done. With the exception of the three or four claims above mentioned, no work of real value has been performed; in very few cases has even the assessment been worked out. Five days' work by four men is deemed sufficient to cover the statutory requirements for one year. In many cases the so-called "assessment-work" is expended in building cabins, or in making roads, and not in the development of the deposit. Hence the present condition of the claims in this district is practically their primeval condition, and the district can boast of but three or four "developed prospects." Of these, the original discovery, now known as the "Morning Star," makes the best showing.

For 1000 feet or more along the side of the hill, and at practically the same elevation, a series of open-cuts or side-hill cuts have been made, exposing the mineralized zone. The "Jack" is found in lenticular masses, nearly parallel to the stratification, sometimes confined to 2 or 3 feet, and again distributed through 10 or 15 feet of rock. "Jack" is rarely seen in mass at the extreme outcrop, but in most of the cuttings carbonates were found at the crop, filling fissures and crevices in the rock, which were usually replaced by "Jack" at a distance of 5 or 10 feet from the extreme outcrop.

In one cutting I saw a solid face of carbonates about 6 feet wide and 10 feet or more high, but it seemed to be merely a land-slide crevice filled with carbonates, perhaps 1, 2 or 3 feet thick. No drifts have been driven in upon the ore on this property; and until this is done it is impossible to make any predictions as to its future. At present it makes a very pretty showing. Two incorporated companies have done some work in this field; the Buffalo Zinc

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Company and the Arkansas Mining and Investment Company. The work done by one was drifting, in a hill one mile east of Rush Creek, on a zone of poorly mineralized rock that evidently did not pay to work. The other company put down a shaft on the Buffalo, at the mouth of Rush Creek, and doubtless struck a promising zone of mineral; at least, so one would judge from the appearance of rock and ore on the dump and in the bins.

What little ore has been shipped from this district was hauled to White river, 6 or 8 miles, floated in flats down to opposite Batesville, 60 or 70 miles (the flats being then poled back by hand), and then hauled 4 or 5 miles to the shipping-point on the railroad; thence by rail to St. Louis. The ore shipped was hand-cobbed or hand-jigged, and the lean ore necessarily went to the dump.

One would naturally assume in advance that such an enterprise could not be financially successful. With proper dressing-plants, treating the low-grade rock and reducing the net cost of concentrates to a minimum, it might be possible to ship by the circuitous route above described; or better still, if reduction-works were added and the metal were shipped direct from the mine, there might be a fair margin of profit, even while still far from railway transportation.

The "Jack" found in this district is remarkably pure. I did not see any galena, or in fact any other metallic ore, and up to the present time, so far as I could learn by inquiry, none has been found. The blende sometimes occurs massive, in strings or irregular lenticular masses nearly or quite parallel to the rock-stratification, and sometimes as "shot-ore," disseminated through the rock in grains. In the shot-ore that I examined the grains of mineral were about $\frac{1}{8}$ to $\frac{1}{4}$ of an inch in diameter, and quite equally distributed, making a very handsome-looking ore. It is hardly necessary to remark that in the absence of dressing-works, the claims showing "shot-ore" have received little attention.

The advantage this district possesses in having deposits located in the hills above water-level is evident. If they prove as extensive and as rich as the prospects indicate, zinc can be made here at very low cost.

No topographic conditions could be conceived better adapted to the cheap exploitation of these nearly horizontal deposits than here exist. We find a table-land of rolling country, sharply cut to a depth of 400 to 600 feet by the Buffalo and White rivers, and at intervals of every mile, or possibly every two or three miles, one of

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their tributaries cutting out a similar cañon or gorge, the course of which is nearly at right angles to that of the main stream. All these tributaries have numerous forks or branches, each of which has eroded its own minor gorge. Thus the horizon of the mineralized zone is repeatedly opened to attack in all these gorges, except, of course, in those not deep enough to cut it. Prospecting for this upper zinc-horizon is very like prospecting for a coal-bed in the Allegheny-Ohio coal-field, where the bed is above water-level and its outcrop runs around into and up every ravine, following closely a water-level contour of the surface.

NOTE ON THE FRICTION OF MINE-CAR WHEELS.

BY R. VAN A. NORRIS, WILKESBARRE, PA.

(Washington Meeting, February, 1890.)

THE following tests were made, during 1889, for the Susquehanna Coal Company, to determine the relative efficiency of several styles of mine-car wheels in use at their collieries at Nanticoke, Pa.

The wheels experimented upon may be divided into two classes: The "old style" (Figs. 1 and 2), which are oiled at every trip; and the "new style" (Figs. 3, 4, and 5), which require oiling about once in two to four months.

Fig. 1 shows a section of a plain cast wheel, with patent annular oil-chamber. This is held in position on the axle by a square, split cotter-pin. Fig. 2 is a similar wheel, with a patent cast bushing, which is readily replaced when worn. The lubrication is effected by a simple oil-hole, but the oil is delivered a little inside of the gauge-line of the wheel, where it is most needed. Figs. 3, 4, and 5 are sections of a "new style" patented, self-oiling wheel. In this, the outer end is cast closed, and the wheel is fastened to the axle by a spring cotter-pin passed through one of the plugged holes. Dirt is prevented from entering the open end by a cap, with faced end and packing-ring, fitting over the end of the hub. The lubricant is introduced into the oil-chamber through one of the plugged holes in the hub.

When the wheel is in motion, the lubricant is thrown by centrifugal force away from the axle, enters the two ports shown, and is

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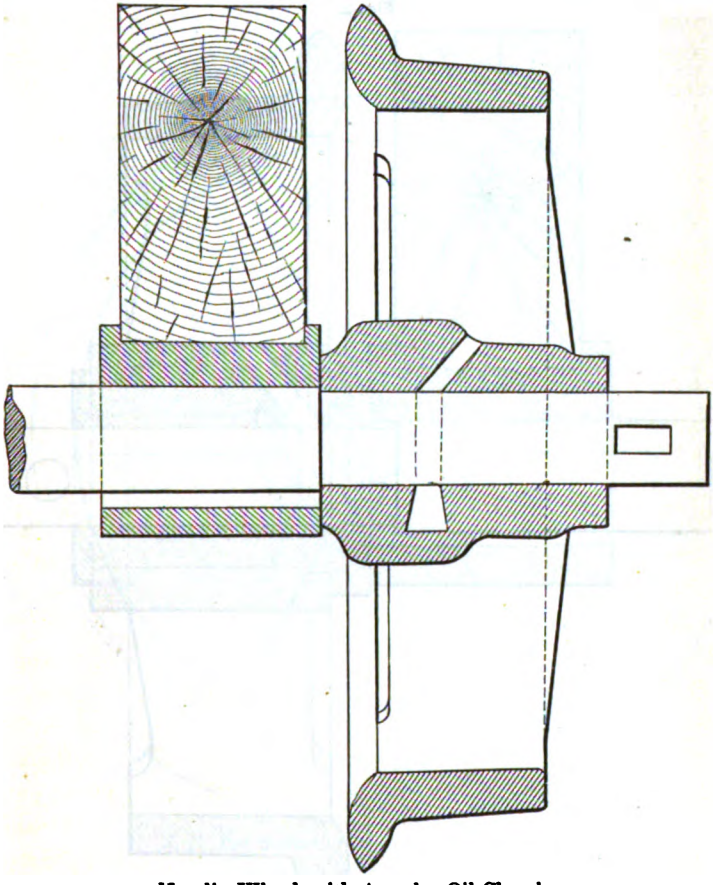
carried through spiral channels into contact with the axle, the surplus oil being carried back to the oil-chamber again. The lubrication appears to be effected by oil adhering to, and being swept back and forth along the axle, in the open zig-zag slit, when passing through the ends of the oil-channels away from the oil-chamber,

Fig. I.



Hamlin Wheel, with Annular Oil-Chamber.

where the section and height of the channel are smallest, the back of the oil-channel being conical with the largest end out, and the radial height of the channel from the axle decreasing as it recedes from the oil-chamber.

All the wheels are of the loose outside type, 16 inches in diameter, mounted on 2⅛-inch steel axles, with journals 5¼ inches long.

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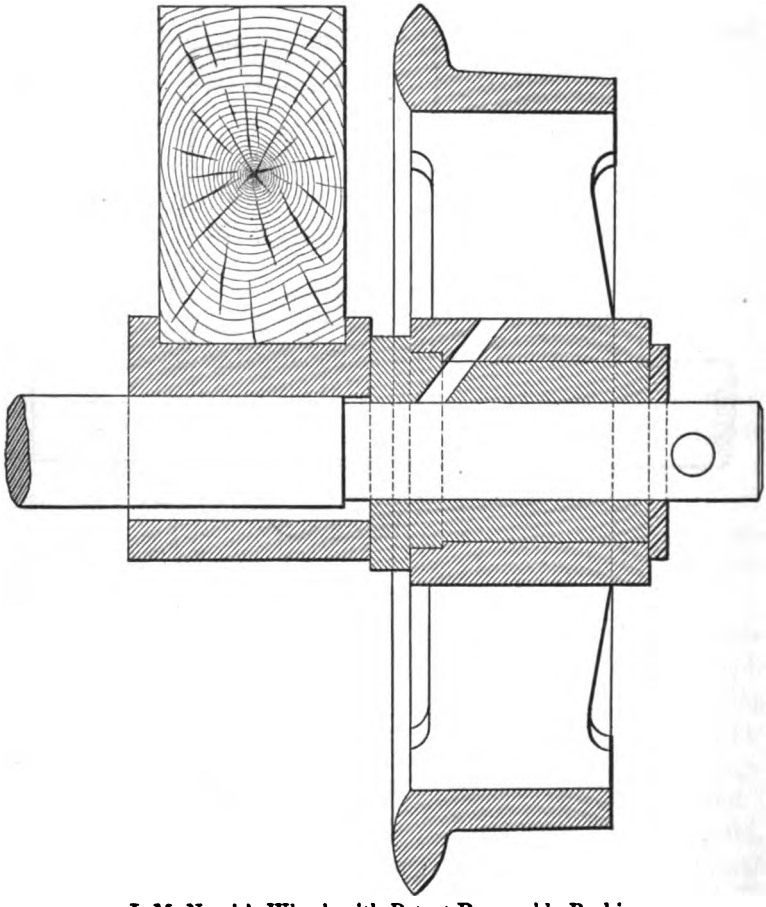
The axles pass loosely through solid cast boxes, bolted to the bottom-sills of the cars, and are not expected to revolve.

Tests were made on the starting and running friction of each style of wheel, under the conditions of empty and loaded cars, level and grade track, curves and tangents. The instruments used were a Pennsylvania Railroad spring dynamometer, graduated to 3000

Fig. 2.



J. M. Norris's Wheel, with Patent Removable Bushing.

pounds, with a sliding recorder, a hydraulic gauge (not recording) reading to 10,000 pounds, graduated to 25 pounds, and a spring-balance, capacity 300 pounds, graduated to 3 pounds. All these were tested and found correct previous to the experiments.

Most of the observations on single cars were made with the 300-

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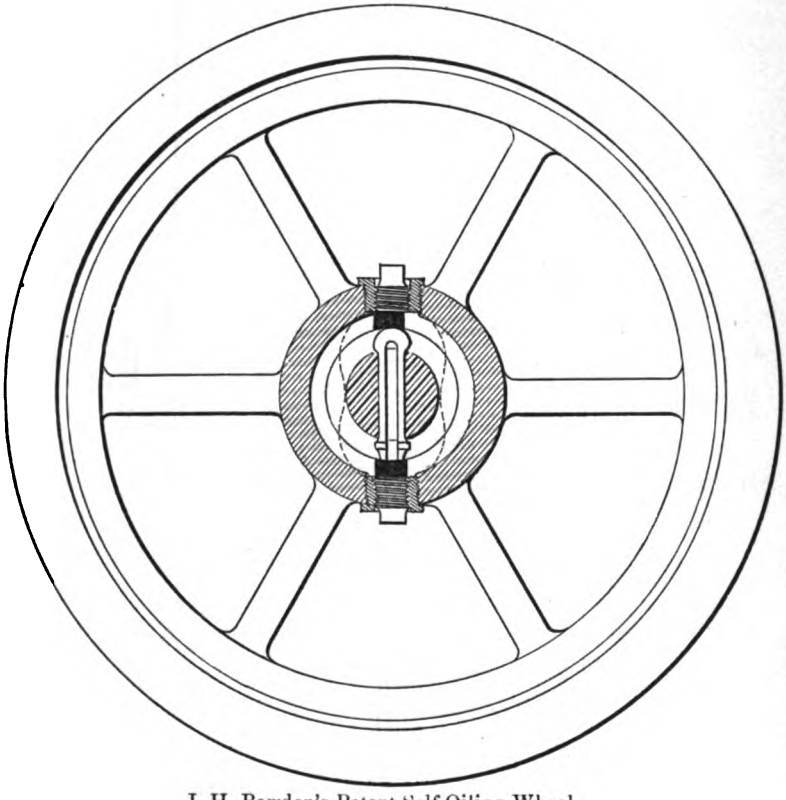
pound balance. The two types of "old style" wheels were found to give results so nearly alike that they have been classed together in the table. Each car was carefully oiled before testing, and several of each type were used, the results being averages from the number of trials shown in the table.

In the experiments upon slow start and motion, the cars were

Fig. 3.



J. H. Bowden's Patent Self-Oiling Wheel.
Section through Hub.

started very slowly by a block and tackle, and the reading was taken at the moment of starting. They were then kept just moving along the track for a considerable distance, and the average tractive force was noted—the whole constituting one experiment.

The track selected for these experiments was a perfectly straight and level piece of 42 inches gauge, about 200 feet long, in rather better condition than the average mine-track. The cars were 41$\frac{3}{4}$-

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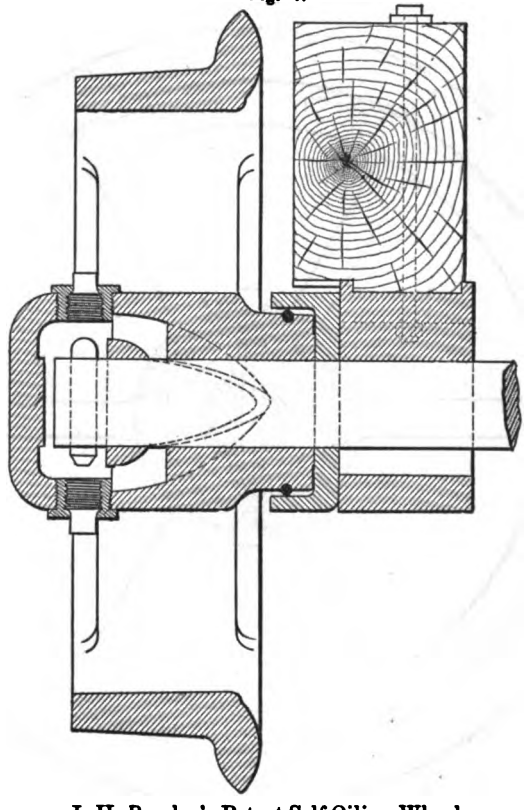
inches gauge, 3½ feet wheel-base, 10 feet long, capacity about 85 cubic feet, with 6-inch topping.

To ascertain the tractive force required at higher speeds, trips of one, four, and twenty cars, both empty and loaded, were attached to a mine-locomotive and run about a mile for each test, the resistance at various points on the track, where its curve and grade were

Fig. 4.



J. H. Bowden's Patent Self-Oiling Wheel.
Section through Oil-Ports.

known, being noted, and care being taken to run at a constant speed. Unfortunately, only four of the "new style" cars were available on the tracks where these trials were made.

The remarkably low results for the twenty-car trips are attributed to variations in the condition of the track, and the fact that the whole train was seldom pulling directly on the locomotive, the cars moving by jerks, so that correct observations were impracticable.

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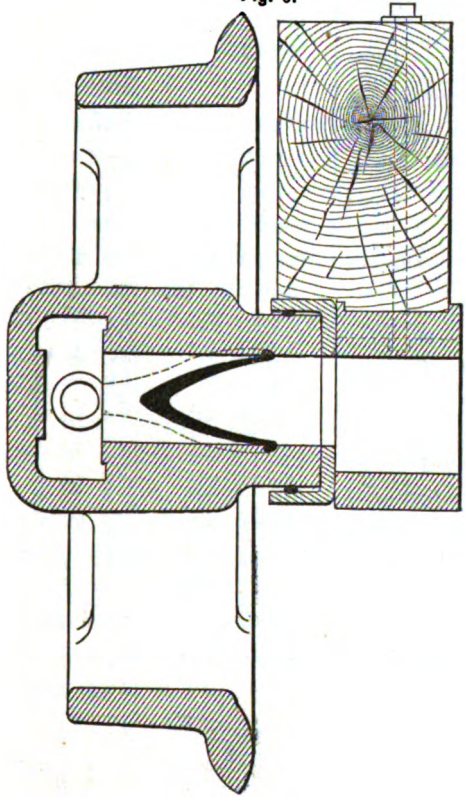
The hydraulic gauge was used for these twenty-car tests, and the needle showed vibrations from one to four tons and back. The mean was taken as nearly as possible. The gauge was rather too quickly sensitive for the work, and the Pennsylvania Railroad dynamometer was not strong enough to stand the starting jerks and the strain of accelerating speed.

The tests marked "rope-haul" were made on an empty-car haul-

Fig. 5.



J. H. Bowden's Patent Self-Oiling Wheel.
Section showing Oil-Grooves.

age-system, about 500 feet long, with overhead endless rope running continuously at a speed of 180 feet per minute, the cars being attached to the moving rope by a chain, a ring at the end of which was slipped over a pin on the side of the car. The increase of friction on the heavier grades was due to the rope pulling at a greater angle across the car. Correction was not made for this angularity at

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SUMMARY OF FRICTION-TESTS ON SUSQUEHANNA COAL CO.'S MINE-CARS, APRIL, 1889.

DIMENSIONS OF WHEELS.

16 inches, diameter of tread.
2¼ inches, diameter of axle.
5¼ inches, length of journal.

OLD STYLE WHEELS.

	Empty.						Loaded.						Number of Tests.			
	Weight of car.	Tractive force per car.	Tractive force due to gr'vity.	Trac. for p'r'car due to frict'n.	Trac. for p'r ton due to frict'n.	Percentage of weight.	Weight of car.	Tractive force per car.	Tractive force due to gr'vity.	Trac. for p'r'car due to frict'n.	Trac. for p'r ton due to frict'n.	Percentage of weight.	Empty.	Loaded.	Empty.	Loaded.
LEVEL.																
Average slow start	2240	100	.....	100	100	4.46	8500	325	.....	325	85	3.80	16	12	6	20
" slow start	2110	83	.....	83	86 8/10	3.88	7885	357	.....	357	101 4/10	4.53	63	53	82	48
" motion 50 ft. per minute	2140	54	.....	54	56 8/10	2.52	7885	205	.....	205	58 2/10	2.60	54	60	81	72
" " 1000 ft. per minute, 1 car	2140	62	.....	62	64 8/10	2.89	7885	262	.....	262	74 1/10	3.32	17	74	6	39
" " " " " " 4 "	2240	62½	.....	62½	62½	2.80							3	.....	6	.....
" " " " " " 20 "	2240	47	.....	47	47	2.20	9000	117	.....	117	29	1.30	6	18		
" starting jerk 20 cars	2240	96	.....	96	96	4.29	9000	175	.....	175	44	1.95	3	7		
" " " 2 " rope-haul	2240	630	.....	630	630	28.12							10			
GRADE.																
" slow start 12°	2140	550	445	105	110	4.90	7885	1950	1639	311	88 4/10	3.94	2	2	2	2
" motion 50 ft. per minute, 12°	2140	510	445	65	68	3.18	7885	1800	1639	161	45 8/10	2.00	10	16	6	12
" " 1000 " " " 1¾° 1 car	2140	125	65	60	62 4/10	2.80							10		6	6
" " " " " " 1½°, 1 "							7885	425	205	220	62 4/10	2.79		12		16
" " 200 ft. per min., rope-haul, 2°	2240	140	78	62	62	2.80							15			
" " " " " " " 2°30'	2240	183	98	85	85	3.80	} Cars pulled from side.						15			
" " " " " " " 5°10'	2240	315	202	113	113	5.00							15			
" " " " " " " 6°10'	2240	353	240	113	113	5.00							15			
CURVE.																
" slow start, 85 ft. radius	2240	125	.....	125	125	5.58	8500	400	.....	400	106	4.70	10	10		8
" " " 11 " " 1½° grade							8700	819	227	592	152	6.80		8		7
" 20 cars 1000 ft. per min., 350 ft. radius	2240	62	.....	62	62	2.80	9000	143	.....	143	36	1.60	5	7		
" 20 " " " " " 450 " "	2240	50	.....	50	50	2.23	9000	126	.....	126	32	1.40	5	7		
" 4 " " " " " 350 " "	2240	100	.....	100	100	4.46							2		2	
													Total number of tests,		276	286	197	280

NEW STYLE WHEELS.

	Empty.						Loaded.						Equals 989 tests total.
	Weight of car.	Tractive force per ton.	Tractive force due to gr'vity.	Trac. for p'r'car due to frict'n.	Trac. for p'r ton due to frict'n.	Perentage of weight.	Weight of car.	Tractive force per car.	Tractive force due to gr'vity.	Trac. for p'r'car due to frict'n.	Trac. for p'r ton due to frict'n.	Percentage of weight.	
LEVEL.													
	2415	66⅔	.....	66⅔	62	2.78	9125	200	.....	200	49	2.20	
	2415	62	.....	62	55 8/10	2.48	8160	193	.....	193	53	2.36	
	2415	40	.....	40	37 1/10	1.66	8160	133	.....	133	36 1/10	1.63	
	2415	36⅔	.....	36⅔	33 8/10	1.48	8160	158	.....	158	43 1/10	1.93	
	2415	37½	.....	37½	34⅔	1.56							
GRADE.													
	2415	600	502	98	90 8/10	4.06	8160	2000	1696	304	83 4/10	3.73	
	2415	540	502	38	35 1/10	1.56	8160	1825	1696	129	35 1/10	1.58	
	2415	100	73	27	25	1.12	8160	400	249	151	41 1/10	1.85	
							8160	350	212	138	37 1/10	1.69	
CURVE.													
							9125	275	.....	275	67	3.00	
							9125	564	239	325	80	3.60	
	2415	75	.....	75	70	3.10							

J. H. BOWDEN, Chief Engineer.

Tests made by R. VAN A. NORRIS, E.M., Assistant Engineer.

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COMPARISON OF RUNNING FRICTION OF CARS WITH OLD AND NEW STYLE WHEELS ON STRAIGHT ROAD, LEVEL AND GRADE, UNDER SIMILAR CONDITIONS. AVERAGE OF 807 TESTS.

	Percentage of Weight.			Comparison of Friction.		
	Old style.	New style.	Difference.	Old style, per cent.	New style, per cent.	Diff. per cent.
Speed, 50 feet per minute.....	2.57	1.61	0.96	100	62 9/10	37 1/10
" 1000 " " "	2.92	1.68	1.28	100	57 7/10	42 3/10

the time, and the rope has since been re-arranged, so that the correction cannot now be made.

There were not enough curve-experiments to permit the deduction of any general formula for the resistance of these cars on curves.

The experiments on grade agree fairly well with those on a level, the rather higher values obtained being probably due more to the greater effort required in moving them and the consequent jerkiness of the motion than to any real increase in resistance. As the experiments on all styles of wheels were made in an exactly similar manner, the comparative value of the results is believed to be nearly correct, the probable error in each set of experiments, as computed by the method of least squares, varying from about 4 per cent. for slow start and motion to 12 per cent. for the rapid motion and twenty-car trips.

The general results, showing an economy in friction of nearly 40 per cent., led to the adoption of the "new style" wheels by the company. These results, obtained as carefully and accurately as the circumstances permitted, are now offered as showing the approximate frictional resistance of such wheels under the ordinary working-conditions.

SOME TESTS OF THE RELATIVE STRENGTH OF NITRO-GLYCERINE AND OTHER EXPLOSIVES.

BY FREDERICK W. CLARK, CHICAGO, ILL.

(Colorado Meeting, June, 1890.)

THESE tests were made under the writer's supervision in the mining and metallurgical laboratories of the Massachusetts Institute of Technology, in the winter of 1888–89, by Mr. Robert C. Williams, of Marquette, Mich., assisted by Mr. J. B. Seager, of the

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same place. The results of the work were fully detailed in Mr. Williams's graduating thesis. The following notes are abstracted from Mr. Williams's thesis, using practically his own words, but omitting many details.

In most of Mr. Williams's tests a very close agreement will be noted in the results of duplicate tests fired under precisely similar conditions. Where there were serious irregularities, the cause was, in most cases, discovered and stated. Many of the duplicates were made with days or weeks intervening.

TABLE I.

Tests of Caps or Exploders.

No.	Name of cap.	Cubic capacity in c.c.		Net enlargement.	Average.	Remarks.
		Bef.	After.			
1	Eagle Triple Face.....	1.10	15.71	14.61	13.75	
2	"	1.15	15 40	14.25		
3	Quintuple.....	1.19	14.25	13.06		
4	"	1.20	14.30	13.10		
5	New York Atlas Quadruple	2.26	10.65	8.39	9.24	Did not fit hole exactly.
6	"	2.10	11.55	9.45		"
7	"	2.05	12.75	10.70		"
8	"	2.15	10.58	8.43		"
9	Eagle Triple Face.....	1.07	8.10	7.03	6.93	
10	"	1.08	7.92	6.84		
11	"	1.18	8.10	6.92		
12	Eagle Special.....	1 25	13.86	12.61	12.46	
13	"	1.31	14.02	12.71		
14	"	1.27	13.32	12.05		
15	San Francisco Cap, Triple Face.....	1.20	12.70	11.50	10 20	
16	"	1.20	9.80	8.60		
17	"	1.25	12.40	11.15		
18	"	1.25	10.80	9.55		
19	German Cap.....	2.25	14.06	11.81	11.88	
20	"	2.25	14.20	11.95		
21	Metallic Cap Co., Newport	2.25	18.6	16.35	14.10	
22	"	2.25	15.1	12.85		

The makers and vendors of explosives readily furnished samples for experiment, but owing to the difficulty of getting powders from a distance, we secured few powders not sold in New York, Boston or Halifax. We are under special obligations to Lieutenant E. L. Zalinski, of Fort Hamilton, N. Y.; to Prof. Munroe, of Newport; and to the Boston agents of the Atlas and Forcite and other powders, who procured the bulk of our samples. A visit to Halifax in the

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spring enabled the writer to procure a few samples of powder made in England, but several of the dealers there refused to allow their powder to be experimented with.

The first requisite in our experiments was to obtain an exploder or cap that we could depend upon to give uniform results, and of sufficient force to explode the various dynamites. We decided to use ordinary double-tape miners' fuse, and tamp with quartz sand.

About 1000 pounds of extra-refined lead was melted at the lowest possible temperature, and cast into shapes convenient for test-pieces. For the cap experiments we used a cylinder $2\frac{1}{2}$ inches high by $2\frac{1}{8}$ inches in diameter, with a slight taper to facilitate removal from the moulds. It weighed about 3 pounds. A $\frac{1}{4}$-inch hole was drilled into it to such a depth that the fulminants in the cap reached to the geometric center of the cylinder. The cap, with a fuse attached, was placed in this hole, and fired in the usual manner. The explosion made a pear-shaped cavity. From a graduated burette water was allowed to run until the cavity was just level-full, when the difference in the reading of the burette gave the exact measure of the cavity in cubic centimeters. Each cavity was measured before and after the explosion.

The Eagle cap, triple and quintuple, was decided upon for the dynamite-tests.

To determine whether the different strengths of caps influenced the result of very small charges of dynamite, the following tests were made, 75 per cent. Forcite powder being used:

TABLE II.

No.	Cap.	Charge.	Cubic capacity in c.c.		Difference.	Average.	Remarks.
			Bef.	After.			
1	Triple	1 gram.	2.08	blowup			Top of cyl. destroyed
2	"	"	2.07	35.26	33.19	32.77	
3	"	"	2.	34.36	32.36		
4	Quadruple	"	1.70	31.25	29.55		Rejected.
5	"	"	1.98	38.86	36.88	36.88	
6	Quintuple	"	1.70	42.12	40.42	40.42	

Allowing for the difference in the caps, there was no appreciable gain in using a stronger cap. In the later experiments, where larger quantities were used, different results were obtained. In small charges the weaker cap develops the full strength of the powder.

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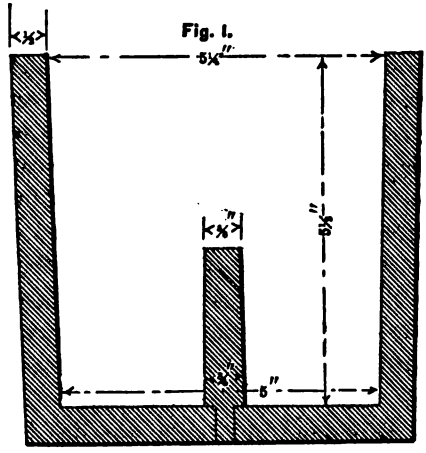
The lead cylinders used for the regular tests were cylindrical, 5¼ inches in diameter and 5½ inches high, and weighed about 20.5 kilos (45.1 pounds). They were cast in a mould having a slight taper to facilitate removal. A steel pin, projecting from the bottom of the mould, 2½ inches long and ⅝ of an inch in diameter, gave a chamber of 14.3 c.c. capacity for the reception of the explosive. (See section, Fig. 1, for further dimensions.)



Section of Mould for Casting Lead Cylinders.

The lead was melted in an iron kettle, care being used to just barely melt the lead, cold lead being added when necessary to obtain the same consistency with every set of castings.

The cylinders made in these moulds were finished ready for charging.

Caps with ordinary double-tape fuse were placed in the cavity and tamped with dry quartz sand to ascertain their effect on the cylinders. The results are shown in Table III.

TABLE III.

No.	Cap.	Charge.	Cubic capacity. Before.	Cubic capacity. After.	Diff.	Average.	Remarks.
1	Eagle Triple.....	1 cap.	14.3	17.	2.7	2.3	
2	" "	"	"	16.3	2.		
3	" "	"	"	16.6	2.3		
4	Eagle Quintuple.....	"	"	17.2	2.9	3.1	
5	" "	"	"	17.5	3.2		
6	Eagle Triple.....	2 caps.	"	21.9	7.6	6.8	
7	" "	"	"	20.4	6.1		
8	Eagle Quadruple.....	"	"	24.0	9.7	9.7	
9			"				

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The next experiment was to ascertain the amount of charge that could safely be fired without rupturing the cylinders. For this purpose a frame was made to hold the lead cylinders firmly when the charge was fired. The charge of dynamite was weighed out on tared tissue-paper, rolled into a compact cylinder, closed at one end; and the fuse, with cap attached, was inserted into the dynamite to the depth of the fulminate. The paper was then drawn tightly around the fuse and fastened with fine copper wire. The charge usually fitted the hole in the cylinder snugly. It was now gently pushed home, and tamped with fine dry quartz sand by means of a wooden tamping-rod. The cylinder is now placed on a wrought-iron plate $1\frac{1}{4}$ inches thick, and a similar plate (with a $\frac{5}{16}$-inch hole in the center for the fuse to pass through) is placed on top. Two wrought-iron rectangular rings of $1\frac{1}{2}$-inch section, made of best Norway iron, are placed around the plates, and wedges are firmly driven between the rings and the upper plate. Fig. 2 is a sketch of the apparatus.

The effect of gradually increased charges is shown in Table IV. Triple force Eagle caps and 75 per cent. Forcite were used.

TABLE IV.

No.	Charge.	Cubic capacity.		Corrected c.c.	Average.
		Before.	After.		
1	2 grams.	14.3	66.24	51.94	54.14
2	2 "	"	70.65	56.35	
3	4 "	"	158.41	144.11	150.01
4	4 "	"	170.22	155.92	
5	6 "	"	281.55	267.25	262.42
6	6 "	"	271.90	257.60	
7	8 "	"	416.	401.70	401.70

A 6-gramme charge was decided to be about as much as the cylinders would stand, without danger of rupture, when the most powerful explosives were used.

The effect of the explosion on the cylinder was barely perceptible on the exterior. On removal from the rings, the upper part of the hole was somewhat enlarged, and a pear-shaped cavity was formed, the broad end being a little below the original bottom of the cavity. The interior of the cavity was usually smooth, and frequently showed that the lead had been momentarily melted, the lines of flow being quite plainly shown. Fig. 3, an Ives' plate from a photograph, shows the nature and extent of deformation.

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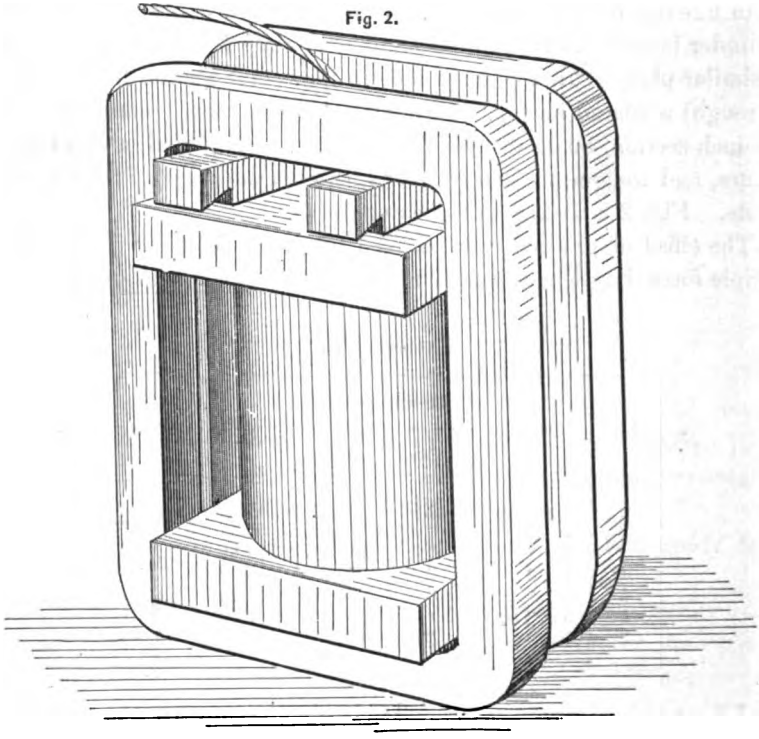
Table V. gives the results in detail.

Remarks on Powders and Tests.—All these tests were made in the same way with but slight modifications. In the case of nitro-glycerine, the liquid was kept dissolved in methyl alcohol, and precipitated out with water as needed. The weighed amount was then placed in a glass tube, and the detonator was placed in a perforated cork, which fitted the tube. The cap extended for about one-third its length into the nitro-glycerine. The tube was then tamped firmly with sand.

Fig. 2.



Cylinder and Frame Ready for Explosion.

In test No. 145 some of the liquid ran over, mixing with the sand, and the result of this test shows a falling off in strength, due to the absorption of the nitro-glycerine in the sand.

In firing gun-cotton, particularly the uncompressed kind, considerable trouble was experienced in getting 6 grammes of it into 14 c.c. of space. Pounding it in with an iron punch resulted, in one case, in the ignition of the charge. The substitution of a wooden punch and a long lever for the hammer was afterwards followed. An enlarge-

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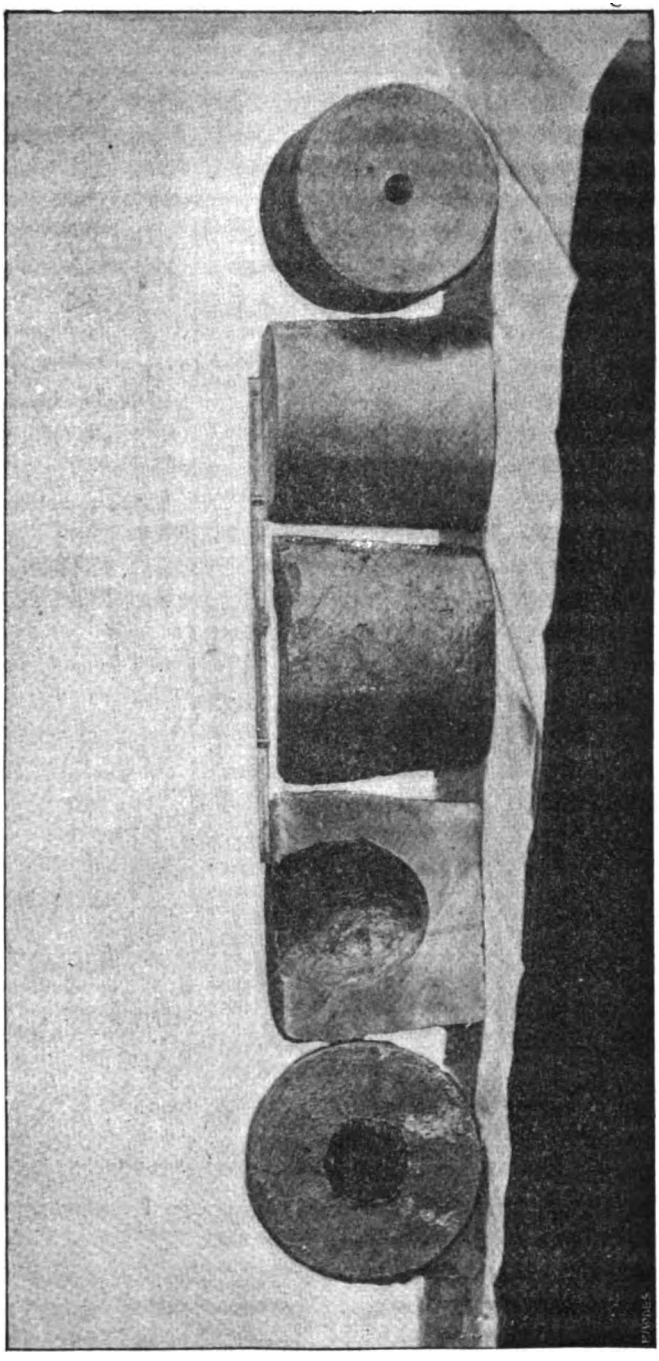

FIG. 8.

Lead Cylinders before and after Explosion, showing Interior and Exterior Deformation.

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TABLE V.

(Triple or Quintuple Eagle Caps used in all Tests.)

Powder.	No. of Test.	Charge, 6 grams Powder used.	Cubic capacity. Bef.	Cubic capacity. After.	Compressed, c. c.	Average compression.	Remarks.
Forcite 30%	17	Triple cap.	14.3	151.2	136.9	131.1	Contained lumps due to imperfect mixing of powder. Dark brown in color.
" "	18	"	"	141.8	127.5		
" "	19	"	"	147.5	133.2		
" "	20	"	"	142 0	127.7		
" "	21	"	"	140.4	126.1		
" "	22	"	"	149.5	135.2		
Forcite 30%	23	Quintuple cap.	"	162.9	148.6	146.8	Showing effect of stronger cap. Quin. Eagle caps used.
" "	24	"	"	159.3	145.0		
Forcite 35%	25	Triple cap.	"	165.0	150.7	133.8	
" "	26	"	"	144.1	129.8		
" "	27	"	"	143 0	128.7		
" "	28	"	"	140.3	126.0		
Forcite 35%	29	Quintuple cap.	"	162.1	147.8	146.6	
" "	30	"	"	159 7	145.4		
Forcite 50%	31	Triple cap.	"	206.5	192.2	190.5	For effect of cap alone, see Table III., p. 4.
" "	32	"	"	202.3	188.		
" "	33	"	"	207.5	193.2		
" "	34	"	"	203.	188.7		
Forcite 50%	35	Quintuple cap.	"	242.1	227.8	216.1	
" "	36	"	"	222.3	208.0		
" "	37	"	"	226.7	212.4		
Forcite 75%	38	Triple cap.	"	288.1	273.8	270.8	Much lighter in color than lower grades and apparently perfectly homogeneous.
" "	39	"	"	289.7	275.4		
" "	40	"	"	277.4	263.1		
Forcite 75%	41	Quintuple cap.	"	325.0	310.7	313.6	
" "	42	"	"	332.5	318.2		
" "	43	"	"	326.2	311.9		
Forcite 75%	154	2 Triple.	"	316.0	301.7	310.4	
" "	155	"	"	333.5	319.2		
Forcite 75%	156	2 Quintuple.	"	362.0	347.7	340.7	
" "	157	"	"	348.0	333.7		
Ætna 35%	44	Triple cap.	"	192.8	178.5	175.1	All the Ætna powders were light in color and homogeneous in appearance.
" "	45	"	"	197.8	183 5		
" "	46	"	"	203.	188.7		
" "	47	"	"	181.7	67.4		
" "	48	"	"	177.9	163 6		
" "	49	"	"	182 9	168 6		
Ætna 35%	50	Quintuple cap.	"	199.9	185.6	186.1	
" "	51	"	"	200.9	186.6		
Ætna 50%	52	Triple cap.	"	236.7	222.4	220.3	
" "	53	"	"	236.6	222.3		
" "	54	"	"	230.3	216.0		
Ætna 50%	55	Quintuple cap.	"	247.5	233.2	237.9	
" "	56	"	"	257.0	242.7		
Ætna 75%	57	Triple cap.	"	238.1	223.8	231.6	
" "	58	"	"	238.8	224 5		
" "	59	"	"	255.0	240.7		
" "	60	"	"	255.1	240.8		
" "	61	"	"	242.5	228 2		
Ætna 75%	62	Quintuple cap.	"	289.8	275.5	269.6	
" "	63	"	"	277.	263.7		

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TABLE V.—CONTINUED.

(Triple or Quintuple Eagle Caps used in all Tests.)

Powder.	No. of Test.	Charge, 6 grams Powder used.	Cubic capacity. Bef.	Cubic capacity. After.	Compressed, c. c.	Average compression.	Remarks.
Atlas 33%.....	147	Triple cap.	14.3	171.7	157.4	160.2	All Atlas powders were medium bro'n color and apparently well mixed.
" "	148	"	"	181.3	167.		
" "	149	"	"	170.6	156.3		
Atlas 33%.....	150	Quintuple cap.	"	178.5	164.2	170.4	
" "	151	"	"	190.8	176.5		
Atlas 40%.....	64	Triple cap.	"	218.2	203.9	203.9	
" "	65	"	"	218.2	203.9		
Atlas 40%.....	66	Quintuple cap.	"	239.5	225.2	227.9	
" "	67	"	"	240.	225.7		
" "	68	"	"	247.2	232.9		
Atlas 75%.....	69	Triple cap.	"	319.8	305.5	302.1	Cracked upper plate, 1¼ (wrought iron.)
" "	70	"	"	315.	300.7		
" "	71	"	"	314 3	300.		
Atlas 75%.....	72	Quintuple cap.	"	340.5	326 2	327.6	
" "	73	"	"	340.3	326.0		
" "	74	"	"	345.	330.7		
Atlas 75%.....	152	2 Quintuple.	"	361.9	347.6	345.9	
" "	153	"	"	358.4	344.1		
Explosive gelatine, 94% camphor'ted, Nobel's formula, made in '87 by Prof. Monroe.	137	Triple cap.	"	149.5	.....	.....	Not well exploded.
	138	"	"	280.0	.....	.....	" "
				253.3	.....	.....	" "
	139	Quintuple cap.	"	355.5	.....	.....	" "
	142	2 Triple.	"	453.5	439.2	439.2	Satisf'ly exploded.
				326.5	.....	.....	Not well exploded.
Nitro-Glycerine...	143	Triple cap.	"	444.3	430.	430.1	Mixed with sand accidentally.
" " ...	144	"	"	444.5	430.2		
" " ...	145	Quintuple cap.	"	336.6	.....	.....	Warmed to about 120° F.
" " ...	146	"	"	476.0	461.7	461.7	
Gun-cotton, loose.	75	Triple cap.	"	305.5	291.2	288.	Obtained from Metallic Cap Co., of Newport, R. I.
" " "	76	"	"	299.	284.7		
" " "	77	"	"	302.5	288.2		
Gun-cotton, loose.	78	Quintuple cap.	"	269.1	254.8	262.3	Ring broken. Fracture apparently crystalline.
" " "	79	"	"	274.5	260.2		
" " "	80	"	"	300.	283.3		
" " "	81	"	"	275.	260.7		
Gun-cotton (gran.)	122	Triple cap.	"	138.	123.7	118.5	Stowmarket. From Prof. Monroe. 15 yrs. old.
... " "	123	"	"	127.6	113.3		
Gun-cotton (gran.)	124	Quintuple cap.	"	210.9	196.6	206.4	Stowmarket. From Prof. Monroe. 15 yrs. old.
" " "	125	"	"	230.5	216.2		
Gun-cotton (comp.)	126	Triple cap.	"	.....	.....	.....	Burnt up. No explosion.
" " "		"					
Gun-cotton (comp.)	127	Quintuple cap.	"	206.7	192.4	201.6	Stowmarket. G. C.
" " "	128	"	"	225.0	210.7		
Gun-cotton (U. S. service).	129	Triple cap.	"	.....	.....	.....	Burnt up.
	130	Quintuple cap.		200.	178.1	178.6	Cavity enlarged to receive charge.
" "	131	"		204.6	179.1		
" " "	132	"	"	241.7	227.4	234.5	
" " "	133	"	"	256.0	241.7		
Hydro-nitro cellu-	134	Triple cap.	"	.....	.....	.....	Not exploded by nitre.
" lose	135	Quintuple cap.	"	191.	.....	.....	
" "	136	"	"	242.7	228.4	228.4	

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TABLE V.—CONTINUED.

(*Triple or Quintuple Eagle Caps used in all Tests.*)

Powder.	No. of Test.	Charge, 6 grams Powder used.	Cubic capacity. Bef.	Cubic capacity. After.	Compressed, c. c.	Average compression.	Remarks.
Rackarock.....	111	Triple cap.	14.3	136.5	122.2	120.5	Chlorate of potash and nitro-benzol. Color dark brown. Odor very disagreeable after mixing.
"	112	"	"	130.5	116.2		
"	113	"	"	137.5	123.2		
Picrate Powder....	114	Triple cap.	"	30.9	16.6	19.8	A green granular substance with a disagreeablesmoke
" " ...	115	"	"	33.8	19.5		
" " ...	116	"	"	37.5	23 2		
Picrate Powder....	117	Quintuple cap.	"	86.7	72.4	68.9	
" " ...	118	"	"	79.7	65.4		
Belemite.....	119	Triple cap.	"			216.1	Not exploded. A yellow finely gran. compound of melo-dynitro-benzol and nitrate of ammon.
"	120	Quintuple cap.	"	225.0	210.7		
"	121	"	"	235.9	221.6		
Nobel's 75%..... (Kieselguhr)	158	Triple cap.	"	174.	159.7	163.2	
" "	159	"	"	178.5	164.2		
" "	160	"	"	180.	165.7		
Nobel's 75%..... (Kieselguhr)	161	Quintuple cap.	"	188.5	174.2	171.1	
" "	162	"	"	182.4	168.1		
Nobel's Gelatine 95% N. G.	163	Quintuple cap.	"	236.0			Not all exploded.
" "	164	2 " "	"	321.0			"
" "	165	" "	"	285.			"
" "	166	" "	"	342.0			"
Forcite Gelatine...	167	2 Quintuple.	"	491.0	476.7	464.0	
" " ...	168	"	"	486.	471.7		
" " ...	169	"	"	458.	443.7		

ment of the hole in the lead cylinder always had the effect of decreasing the size of the cavity formed by the explosion. Gun-cotton seems to produce a more rending explosion than does dynamite. Several rings were broken and plates indented and split in the few tests we made, while the effect on the lead was less than in many of the dynamites. This is partially accounted for by the absence of tamping, or the greatly reduced amount that was used.

It is evident that gun-cotton and gelatine dynamite require very strong caps—much stronger than those ordinarily used for dynamite. From the variation in our tests, it is probable that the caps used did not fully develop the strength of the gun-cotton. In the case of gelatine dynamite, in only one instance does the quintuple cap appear to develop the full strength of the powder. After some of the powder

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tests, small particles of the gelatine were found, and in remelting the lead cylinders used, incipient explosions occurred.

In Table VI. are shown, in condensed form, the results of the previous table. Assuming nitro-glycerine as a standard of 100, the comparative values of the other powders are deduced:

TABLE VI.

Powder.	Triple Cap.			Quintuple Cap.		
	Enlargement in c.c.	Due to cap.	Comparative strength.	Enlargement in c.c.	Due to cap.	Comparative strength.
Nitro-Glycerine	430.1	2.3	100.00	430.1	2.3	100.00
Forcite, 30%	131.1	"	30.5	146.8	3.1	34.14
" 35"	133.8	"	31.1	146.6	"	34.13
" 50"	190.5	"	44.1	216 1	"	50.25
" 75"	270.8	"	63.0	313.6	"	72.92
Ætna, 35%	175.1	"	40.7	186.1	"	43.27
" 50"	220.3	"	51.25	237.9	"	55.3
" 75"	231.6	"	53.85	269.6	"	62.7
Atlas, 33%	160.2	"	35.00	170.4	"	39.6
" 40"	203.9	"	47.4	227.7	"	52.94
" 75"	302.1	"	70.24	327.6	"	76.17
Gun-Cotton (loose)	288.0	"	67.00	262.3	"	61.00
" (granulated)	Not Ex.		27.54	206.4	"	48.23
" (compressed)	"		.....	201.6	"	46.88
" (U. S. Service)	"		.....	234 5	"	54 5
Forcite Gelatine	.....	.....	.....	464.0	"	107.55
Nitre Cellulose	"		.....	228.4	"	53.1
Explosive Gelatine	"		.....	439.2	"	102.1
" " Nobel's	"		.....	.....	.....	Not Ex.
Rackarock	120.5		28.00	.....	"	
Picrate	19.8		4.6	68.9	"	16.1
Belemite	.....	.....	.....	216.1	"	50.25
Nobel, 75%	163.2		37.9	171.1	"	39.8
Nitro-Glycerine (warm)	461.7		107.34	.....	.....	.....

Conclusions.—It will be observed, in looking over these results, that in very few cases is the result higher than should result from the nitro-glycerine contents, but in some cases it is lower than the nitro-glycerine percentages would lead one to expect. Furthermore, the quintuple cap almost invariably gives a better result than the triple force cap, and two caps give a better result than one. Also, in the case of some of the samples of gun-cotton, and particularly in the cases of two samples of gelatine dynamite, a perfect explosion could not be obtained even with two quintuple caps. Evidently, when these powders are used, a special cap or primer is required.

In the case of Nobel's kieselguhr dynamite, the explosion seemed to be perfect, but the results are far below what they should be if we

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judge by the percentage of nitro-glycerine present. Our tests lead us to the belief that the low results are due to the retarding effect of the inert absorbent. It is all but universally believed that an explosive dope or base greatly increases the force of explosion, and we believe it true in the sense that if the base is so compounded as to take care of itself, it will allow the full force of the nitro-glycerine present to be exerted, while a base not thus compounded will act as a drag on the nitro-glycerine. All the dynamites with explosive base approach, in nearly every case, the strength represented by their nitro-glycerine contents, in a few cases exceeding it (in some of the lower-grade powders), owing probably to superior chemical properties of the absorbent. Only one of the 75 per cent. powders (Atlas) shows a strength exceeding 75 per cent. of nitro-glycerine, while Ætna shows a decided falling off in strength. The lower grades of this powder were exceptionally good.

In Drinker's tests with a mortar, he shows that nitro-glycerine, when mixed with black powder, gives a power greatly exceeding the sum of the intensities generated by each separately. From Mr. Williams's tests it does not seem that this is possible, especially to such a marked degree, and he accounts for it as follows: Mr. Drinker found that a charge of black powder barely lifted the ball; that the nitro-glycerine alone threw it 25 feet, and that the mixture of the two threw it 75 feet. Now, nitro-glycerine is very quick acting, while black powder is comparatively slow. The ball only rests on the mortar bore, and, if raised the slightest amount, would allow the gases to escape. It is probable that the force of the nitro-glycerine is not wholly spent in lifting the ball, but, being so quick, it raises the ball, and the major part of the force is dispersed into space before the ball, in consequence of its inertia, could receive its full effect. On the other hand, when nitro-glycerine is mixed with powder, a slower combustion takes place, and a greater projectile force is exerted on the ball, and less lost by dispersion. The nitro-glycerine exerts a rending or compressive force which is necessarily mostly lost, while the intensity of the slower-acting mixtures is expended more in projectile force, which raises the ball.

A specific-gravity test made of one of the lead cylinders before and after firing, resulted as follows:

	Kilos.
Weight of cylinder in air, ...	19.72
Weight of cylinder in water, ...	17.97
Weight of displaced water, ...	1.75

19.72 ÷ 1.75 = 11.268 specific gravity after firing.

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Before firing, the specific gravity was found to be 11.51, showing a rather unlooked for result, *i.e.*, the lead was less dense after than before firing.

Tests of Black Powders.—A few of these were tried in the same way as dynamite powders, with fuse and cap, and with cap only. The statement has been made on good authority that black powder fired with a detonator is much more powerful than when fired by

TABLE VII.

Tests of Black Powders.

Name of Powder.	No. of test.	Charge.	Fired by	Contents.		Compressed c.c.	Average.	Remarks.
				Before firing, c.c.	After firing, c.c.			
Hazard Sporting.....	82	6gm.	Fuse	14.3	34.4	20.1	21.3	60 cents per lb. powder.
" "	83	"	"	"	36.5	22 2		
" "	84	"	"	"	36.0	21.7		
Hazard Sporting.....	85	"	Triple	"	34.0	19.7	15.1	
" "	86	"	cap and	"	28.2	13.9		
" "	87	"	fuse.	"	25.8	11.5		
Hazard Sporting.....	88	"	Quin.	"	34.8	20.5	17.4	
" "	89	"	cap and	"	28.8	14.5		
" "	90	"	fuse.	"	31.4	17.1		
Oriental Sporting.....	91	"	Fuse	"	27.2	12.9	13.2	60 cents per lb. powder.
" "	92	"	"	"	27.7	13.4		
Oriental Blasting.....	93	"	"	"	25.6	11.3	11.3	9 cents per lb. powder.
" "	94	"	"	"	25.6	11.3		
Laflin & Rand Sporting	95	"	"	"	33 4	19.1	19.4	90 cents per lb. powder.
" "	96	"	"	"	34 0	19.7		
Laflin & Rand Blasting	97	"	"	"	20.9	6.6	13.7	9 cents per lb. powder.
" "	98	"	"	"	35.2	20.9		
American Sporting.....	102	"	"	"	31.2	16.9	15.5	90 cents per lb. powder.
" "	103	"	"	"	28.3	14.0		
American Blasting.....	104	"	"	"	29.4	15.1	15.1	11c. lb. powder.
Du Pont Sporting.....	105	"	"	"	28.9	14.6	15.9	90 cents per lb. powder.
" "	106	"	"	"	31.6	17.3		
Du Pont Blasting.....	107	"	"	"	20.	5.7	5.4	11 cents per lb. powder.
" "	108	"	"	"	19.3	5.		
Du Pont Sporting.....	109	"	"	"	30.7	16.4	15 8	60 cents per lb. powder.
" "	110	"	"	"	29.4	15.1		

fuse alone. The opposite was found to be true in our tests. The powders were well tamped in all cases. The powders of different makers were selected as nearly alike as possible in grain and price.

The tests made are not satisfactory on account of the gases from

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the slow burning powder dissipating themselves into space. The tamping and fuse were often thrown out with great force, as from a gun, while little effect would be produced on the lead cylinder. Our tests, then, merely show the rupturing or rending strength of the powder with dry tamping in a short length of tube, while no measure of the projectile force is obtained. In the higher-grade dynamites there was apparently very little projectile force; with the lower-grade dynamites there was more; and in the black powders it was very marked.

Table VII. exhibits the results of the tests made with the Hazard, Oriental, Laflin & Rand, American and DuPont powders; the samples of the first being sporting powders only, while of all the rest, blasting as well as sporting powder was tested. The final column gives the price per pound of each of these explosives.

Messrs. Williams and Seager worked industriously for about four months on the preceding tests. This manner of testing gives consistent results where the powders tried are homogeneous and well mixed, and if tried with larger lead cylinders, one-ounce charges or more, and a greater variety of powders, the results ought to be very valuable.

THE PROPERTIES OF ALUMINUM, WITH SOME INFORMATION RELATING TO THE METAL.

BY ALFRED E. HUNT, JOHN W. LANGLEY AND CHARLES M. HALL, PITTSBURGH, PA.

(Washington Meeting, February, 1890.)

The Purity of Commercial Aluminum.

A great deal that has been written heretofore about the properties of aluminum is of doubtful value, owing to the lack of knowledge we have of the purity of the aluminum referred to. Much of the metal heretofore sold in the markets as pure aluminum has been contaminated with from 4 to 10 per cent. of impurities. Indeed, it is only within a very few years that a purer metal than 96 per cent. aluminum has been made upon any larger than a laboratory scale.

The works of M. Pechinet, of the Société Anonyme d' Aluminium, at Salindres (Gard), near Marseilles, in France, have long enjoyed the

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reputation of making the purest aluminum placed upon the market. The method employed is essentially Deville's, reducing aluminum chloride by aid of metallic sodium, as practiced also by the Aluminium Company, Limited, at Oldbury, in England, and (with the modification of using aluminum fluoride instead of aluminum chloride, to be reduced by the sodium) by the Alliance Aluminium Company, of Wallsend-on-Tyne, in England.

The Pittsburgh Reduction Company, manufacturing under the Hall process, reduces the metal from its oxide, alumina (Al_2O_3), by electrolysis, this alumina being held in solution by a molten fluoride bath, which itself is not decomposed by the electric current. The latter is conveyed to the melted solution by means of carbon cylinders placed in the bath for positive electrodes, a carbon-lined metal pot forming the negative electrode. The oxygen of the alumina goes off at the positive electrode as carbonic acid, wearing away the carbon at the rate of nearly a pound of carbon to the pound of aluminum produced. The reduced metal settles to the bottom of the pot, from which it is easily tapped or ladled off, practically free from the electrolyte. Remelting entirely purifies it.

All four of these concerns have succeeded, although only with careful selection of materials, and the greatest care to prevent contaminations, in making aluminum of over 99 per cent. purity, and in regularly placing upon the market in considerable quantities metal of over 98 per cent. purity. They are the only establishments known to us which have done so.*

The impurities of the metal made by the sodium process, as practiced by the first three, have been nearly half iron and half silicon. In the case of the Pittsburgh Reduction Company the impurity consists almost entirely of silicon, the iron being less than ten-hundredths of 1 per cent.

In the first three-quarters of the month of December, 1889 (the plant was shut down for repairs, the first time since November 28, 1888, from the day before Christmas until New Year's), 1250 pounds of aluminum was made by the Pittsburgh Reduction Company, *all* of which contained over 98.25 per cent. aluminum, with less than 0.10 per cent. iron, and with about 1.50 per cent. silicon. A con-

* Since this paper was first printed, we have received a communication from the Swiss Company at Neuhausen, protesting against the assumption that its product is fairly represented by the analysis given in Table I. This analysis was made upon a sample obtained at Neuhausen by Mr. Hunt, who saw it cast. We do not, however, insist upon it as a fair average; and we shall be happy to publish the analysis of any further sample.

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siderable quantity of this metal contained over 98.75 per cent. aluminum.

A singular fact is that no small portion of the silicon in aluminum exists in the graphitoidal state. This was pointed out by Mr. H. J. Williams, then chemist of the Pittsburgh Reduction Company, in *Transactions*, vol. xvii., page 542, and was recognized by Deville also in his researches. In aluminum containing 6.20 per cent. of silicon, made by the Alliance Aluminium Company, Limited, we have found 3.85 per cent. of graphitoidal silicon. In a sample of aluminum made by the Pittsburgh Reduction Company from calcined native bauxite, and containing 5.30 per cent. of silicon, 2.40 per cent. existed in the graphitoidal state. This graphitoidal silicon can readily be separated by dissolving the metal in dilute hydrochloric acid, adding enough bromine water to keep the solution well oxidized, and to prevent loss as silicuretted hydrogen. (Boron, if added to aluminum, is found to exist similarly in the metal in the dual form of graphitic and diamantine boron.—*Deville.*)

The following table gives the average composition of the metal of the various makers, of given percentages of contained aluminum, as shown in many analyses made by the Pittsburgh Testing Laboratory:

TABLE I.

Composition of Various Commercial Aluminums.

Aluminum metal made by	Per cent. of aluminum.	Per cent. of combined silicon.	Per cent. of graphitoidal silicon.	Per cent. of iron.	Per cent. of copper.	Per cent. of sodium.	Per cent. of lead.
The Pittsburgh Reduction Company	95.00	1.50	1.35	2.00	0.07	0.04	0.03
The Société Anonyme d' Aluminium	95.00	0.90	1.00	3.00	0.05	0.01	0.04
The Alliance Aluminium Company, Limited	95.00	0.90	0.75	3.25	0.02	0.07	0.01
The Aluminium Company, Limited	95.00	0.85	0.75	3.00	0.10	0.30	nil.
The Aluminium-Industrie-Aktien-Gesellschaft, of Neuhausen.	95.00	1.75	1.15	2.00	0.01	0.07	nil.
	94.15	1.02	0.54	2.80	1.49	.....	.....
The Pittsburgh Reduction Company	97.00	1.55	1.25	0.13	0.03	0.02	0.02
The Société Anonyme d' Aluminium	97.00	0.90	0.82	1.20	0.04	0.01	0.03
The Alliance Aluminium Company, Limited	97.00	0.95	0.58	1.45	0.01	0.05	0.01
The Aluminium Company, Limited	97.00	0.75	0.52	1.55	0.03	0.15	nil.
The Pittsburgh Reduction Company	98.00	1.30	0.60	0.07	0.01	0.01	0.01
The Société Anonyme d' Aluminium	98.00	0.71	0.85	0.90	0.01	0.01	0.02
The Alliance Aluminium Company, Limited	98.00	0.90	0.29	0.75	0.01	0.04	0.01
The Aluminium Company, Limited	98.00	0.90	0.23	0.80	0.02	0.05	nil.
The Pittsburgh Reduction Company	98.52	0.42	0.72	0.05	0.06	nil.	0.04
" " " "	99.00	0.80	0.15	0.03	0.01	nil.	0.01
The Société Anonyme d' Aluminium	99.00	0.35	0.13	0.50	nil.	0.01	0.01
The Alliance Aluminium Company, Limited	99.00	0.31	0.20	0.45	nil.	0.03	0.01
The Aluminium Company, Limited	99.00	0.27	0.15	0.55	0.02	0.01	nil.
The Pittsburgh Reduction Company	99.20	0.41	0.34	0.05	nil.	nil.	nil.
The Alliance Aluminium Company, Limited	99.14	0.23	0.17	0.46	nil.	nil.	nil.
The Pittsburgh Reduction Company	99.34	0.40	0.21	0.05	nil.	nil.	nil.

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A sample lot of aluminum made by the Pittsburgh Reduction Company, taken to represent as fairly as possible the average metal made by this company, was submitted to many physical tests, which are reported in various parts of this paper. For the purpose of identification and brevity, this metal will be referred to as "the average lot 98.52 per cent. aluminum."

This metal had the following composition:

	Per cent.
Aluminum,	98 52
Silicon, combined,	0.42
Silicon, graphitic,	0.72
Iron,	0 05
Copper,	0.06
Lead,	0.04
Arsenic,	none.
Sulphur,	none.
Phosphorus,	none.
Calcium,	trace.
Sodium,	trace.

Properties With Reference to Specific Gravity.

One of the most striking properties of aluminum is its lightness, which, for many purposes, makes it comparable in value with metals costing one-fourth as much.

The following table gives the specific gravities of aluminum, with authorities:

Table II.

Determinations of the Specific Gravity of Aluminum.

Quality, make and condition of the aluminum.	Specific gravity compared with water at 4° C.	Authority.
Cast metal	2.56	Roscoe and Schorlemmer.
Hammered metal	2.67	" " "
Cast metal	2.60	Guettier's *Metallic Alloys.*
Heated to 100° C and cooled	2.65	Richards' *Aluminum.*
Cast metal absolutely pure at 4° C	2.60	Deville.
" " " " "	2.583	Mallet.
Sample sawed out of center of cast ingot ¾"x6"x12"	2.587	Pittsburgh Testing Labor't'y.
Same annealed	2.692	" " "
Aluminum casting 92.57 per cent. Al 4.05 per cent. Fe 3.38 per cent. Si	2.807	" " "
Aluminum rolled bar 2"x1"x¼" Al 98.00, Fe 0.45, Si 1.51 per cent	2.761	" " "
Average lot 98.52 per cent. aluminum, The P. R. Co., castings	2.610	" " "
Average lot 98.52 per cent. aluminum, The P. R. Co., sheet of .0625 thick	2.710	" " "
Average lot 98.52 per cent. aluminum, The P. R. Co., casting ¾" thick	2.587	John W. Langley.
Average lot 98.52 per cent. aluminum, The P. R. Co., casting ¾" round rod	2.708	"

The specific gravity of aluminum being taken as one, soft steel i

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very nearly 2.95 times, copper 3.6 times, ordinary high brass 3.45 times, nickel 3.5 times, silver 4 times, lead 4.8 times, and gold 7.7 times as heavy.

A sheet of aluminum 12 inches square and 1 inch thick weighs 14.03 pounds; a bar of aluminum 1 inch square and 12 inches long, 1.17 pounds; a bar of aluminum 1 inch in diameter and 12 inches long, 0.918 pounds.

	Pounds.
A cubic inch of cast aluminum weighs,	0.092
A cubic foot of cast aluminum "	158.967
" soft steel "	490.450
" wrought iron "	485.874
" copper "	554.988
" ordinary brass, "	524.160

Aluminum has about the tensile strength of cast-iron, with only about one-third of its weight, and may be cast as easily and successfully. It will therefore be advantageously used to replace cast-iron in the parts of moving machinery that have to be reversed or otherwise have their momentum overcome; since, at one-third the weight, they will have only one-third of the momentum.

Action of Heat.

Pure aluminum melts and becomes fluid at about 1200 degrees Fahrenheit. The amount of impurity in aluminum materially alters its melting-point. One per cent. of iron raises the melting-point over 100 degrees. It does not, like lead, remain firm almost to the fluid-point and then suddenly give way, but passes through a stage from 1000 to 1200 degrees Fahrenheit, in which the metal becomes pasty, loses much of its power of cohesion, and, if it be gently pressed together, can be readily welded. It is, however, very red-short at this temperature, and will not stand hammering to weld the metal without crumbling down. If the metal is not held too long a time in this pasty condition, it does not seem to have been injured after being again cooled down.

The Thomson Electric Welding Company has no difficulty in welding aluminum satisfactorily, rapidly and cheaply, and so firmly that the welded strips will break in the body of the bar. This is done in the admirable Thomson machine, by which the surfaces to be welded are brought together at the welding-temperature (near that of fusion, for many metals, and a temperature at which the rigidity is almost entirely lost in all of them) automatically, in such a way that the welded faces will cohere without changing the form of the constituent parts.

The melting-point of aluminum, as given by Roscoe & Schor-

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lemmer and by Mierzinski, is 700 degrees C., or 1292 degrees Fahrenheit; by Deville, at a temperature higher than zinc and lower than silver; by Fahquhardt, at 760 degrees C., or 1400 degrees Fahrenheit. Mallet says that pure aluminum is more infusible than impure.

Aluminum, in a compact mass, is an absolutely "fixed" metal at any temperature attainable by the combustion of carbon in the most approved furnace, although in thin-beaten foil aluminum burns in a current of oxygen gas. An experiment made by one of the authors by heating a weighed amount of aluminum in a carbon-lined crucible for several hours in a Sefstrom furnace, at a temperature considerably above the melting-point of steel, showed that the resulting buttons of aluminum contained all the original aluminum melted, plus several per cent. of carbon, which formed a tenacious, closely adhering skin upon it. The metal, however, had become quite brittle and had largely lost its cohesive powers. Deville, Watts and Fremy all say that aluminum is absolutely fixed at all temperatures, and loses no part of its weight when highly heated.

Aluminum requires no flux to assist its fusion or form a covering of slag over it. Indeed, most fluxes are absolutely injurious, in that they assist the metal in absorbing silicon and iron from the lining walls of the containing crucible. Charcoal or other light substances, which are sometimes used to cover the molten bath, are very difficult to keep from contaminating the castings, and are of no use whatever. Molten aluminum takes a very thin coating of oxide of aluminum on its surface, which seems to protect the liquid metal underneath it. This thin coating can be discovered by drawing a rod of aluminum across the surface of the bath, and noting the brightened surface of the metal in the streak made by the rod, parting the surface-film. It is not our experience that this film of oxide is of serious consequence in preventing sound castings, or causing flaws in rolled sheets or bars.

Aluminum had best be melted in first-quality plumbago crucibles, from which it absorbs only about 0.25 per cent. of silicon with each remelting. When melted in clay, Battersea or other siliceous crucibles, aluminum becomes very seriously contaminated with silicon.

For purposes rendering purity especially desirable, it is best to remelt in crucibles brasqued or lined with a mixture of finely-ground, "dead-burned," calcined, pure magnesia, with just enough boiled tar to give it sufficient plasticity. This mixture entirely prevents the molten aluminum from coming in contact with the silica in the clay of the crucible walls.

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Heated in an atmosphere of chlorine gas, aluminum burns to a chloride; moist chlorine gas, even in the cold, acts energetically upon it. It is most malleable between 200 and 300 degrees Fahrenheit, although it can be rolled cold with frequent annealing. It should not be worked at a temperature above 400 degrees Fahrenheit, for it becomes very red-short at a temperature a little above this point. It is much hardened by work upon it, either by cold-rolling, hammering or drawing. Drawn through wire-dies, it assumes over double its normal tensile strength, although it does not, like steel, lose proportionately as much of its "ductility," as measured by the flow of the metal in reduction of area at point of fracture.

Like the precious metals, it requires frequent annealings to prevent its cracking during rolling or drawing. This is accomplished by heating the metal to about 800 degrees Fahrenheit, at which heat a thin bar of iron, placed in the muffle, will just appear slightly red on a dark day or at twilight. The aluminum at this temperature should not appear at all red. This temperature can be determined in a practical way by drawing a soft pine stick across the surface of the aluminum to be annealed; it should leave a black mark from the charring of the wood, which should burn off very slowly or not at all. After being thus heated, the metal should be allowed to cool off very gradually. A satisfactory annealing of light articles can be attained, however, by plunging them into water at between 60 and 100 degrees Fahrenheit. When it is necessary to keep the heat down below the point where the metal would sag or lose its shape, annealing can be done by heating the articles in boiling linseed oil, and allowing them to cool gradually with the oil. Very thin sheets and wire can be annealed by plunging into boiling water and allowing to cool with the water. Annealed aluminum is very soft and pliable indeed; in fact, as compared with its tenacity, it is probably the most pliable of metals.

Aluminum, on slowly cooling from fusion, crystallizes in octahedrons; and castings, on fracture, show a fine-grained crystalline structure. Pulled in a testing-machine, the fracture is granular.

An ingot of aluminum high in silicon was melted in a plumbago crucible, and one portion cooled in a very cold and thick-walled ingot-mould into a very thin ingot, in a way to give the metal as much of a chill as possible; the other portion of the metal was cast in a hot mould and into a thick ingot. The metal of the outside of each ingot was analyzed at the Pittsburgh Testing Laboratory with the following results:

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TABLE III.

Effect of Slow and Rapid Cooling.

	Metal chilled rapidly.	Metal cooled slowly.	Original metal made.
	Per cent.	Per cent.	Per cent.
Aluminum	96.55	96.52	96.60
Iron	0.24	0.24	0.24
Combined or amorphous silicon	1.08	1.34	1.24
Graphitoidal silicon	2.13	1.90	1.92

If it be safe to generalize from this single experiment, graphitoidal silicon is not rendered amorphous or combined to any considerable extent, as would be the case with carbon in chilled iron under the same conditions. This view is supported by the fact that no appreciable difference in hardness can be determined between surfaces of ingots cast in chills and those cast in hot moulds.

The specific heat of aluminum is 0.2143, water being taken as 1.0000, as given in Richards's *Aluminium*. Mallet gives the specific heat as 0.2253; Fremy, as 0.2181. Aluminum follows the general rule of specific heats—that they are inversely as the atomic weights.

The coefficient of linear expansion, as determined by one of the authors of this paper, on rolled rods, $\frac{5}{8}$-inch in diameter, of aluminum of the "average lot" Pittsburgh Reduction Company of 98.52 per cent. aluminum described on page 531, is approximately .0000206 per degree centigrade between the freezing- and boiling-points of water, that of iron by the same method being .0000122. From this it is apparent that aluminum has a coefficient of expansion closely approaching that of tin, which is .0000217, and higher than copper, which is .000017182 (the latter two being quoted from Ganot). Multiplying this factor by $\frac{5}{9}$, to reduce it to the Fahrenheit basis, we have 0.0000115, or 0.00115 feet on 100 feet of aluminum, for a rise in temperature of 1 degree Fahrenheit from 32 degrees.

Sound castings of aluminum can be readily made in dry sand moulds, which can be advantageously lined with plumbago. The metal should be poured quickly, and at very little above the melting-point; otherwise the castings will be unsound. Molten aluminum flows readily, and not much larger gates are needed than for brass.

The shrinkage of average aluminum castings made by the Pittsburgh Reduction Company is about $\frac{17}{64}$ inch to the foot, as measured by casting in a sand mould between two fixed steel points; the shrinkage can be reckoned to be ordinarily about 2.26 per cent. of the length of the mould.

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Thermal Conductivity.

We find the thermal conductivity to be, very nearly, inversely as the electrical resistance. Unannealed wire of "the average lot 98.52 per cent. aluminum," at 14 degrees C., has a conductivity of 17, where copper has a thermal conductivity of 32, the ratio being as 1 to 1.88. Prof. Carhart has made the following further accurate determinations at the University of Michigan by the method of Wiedemann and Franz, taking silver as 100 and copper as 73.6.

Unannealed Aluminum of "The Average Lot 98.52 per cent. Aluminum."

Experiment No. 1. Coefficient of thermal conductivity,	K = 37.83
" " 2. " " "	K = 37.68
" " 3. " " "	K = 37.94
" " 4. " " "	K = 37.63
" " 5. " " "	K = 38.15
Mean of the five experiments,	K = 37.84

Annealed Wire of the Same Lot.

Experiment No. 1. Coefficient of thermal conductivity,	K = 39.32
" " 2. " " "	K = 38.68
" " 3. " " "	K = 38.68
" " 4. " " "	K = 38.80
Mean of the four experiments,	K = 38.87

The annealed metal is distinctly a better heat-conductor, as shown by all the experiments, than the unannealed.

The following table of relative heat-conducting powers is quoted from Ganot's *Physics*, with the insertion of aluminum as above determined by Prof. Carhart:

Table IV.

Relative Thermal Conductivities.

	K.
Silver,	100.00
Copper,	73.60
Gold,	53.20
Annealed aluminum,	38.87
Unannealed aluminum,	37.84
Tin,	14.50
Iron,	11.90
Steel,	11.60
Lead,	8.50
Platinum,	8.40
Rose's alloy,	2.80
Bismuth,	1.80

Corrodibility.

Aluminum becomes covered by a very thin, almost imponderable, coating of oxide on its surface exposed to the atmosphere, which

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seems to protect it from further oxidation. This coating is so thin as hardly to interfere in any way with the surface-polish which the metal takes, and does not change the weight of the metal as determined by the most delicate of chemical balances.

In the chemical laboratory of the Pittsburgh Testing Laboratory we have a thin weighing-scoop, hammered out of an aluminum sheet, which weighed 2.2086 grammes eight years ago; it has not changed in weight one-tenth of a milligramme since.

The popular general statement that aluminum is not acted upon by air, either moist or dry, or by water, is practically true. According to Deville, water does not act on aluminum either at ordinary temperatures or at the boiling-point. The accuracy of this statement the authors have frequently verified, finding aluminum wire, subjected to the action of steam and heated air, to retain its original polish, and not to lose weight, after six hours' exposure.

Aluminum containing sodium is rapidly acted upon by hot water, dissolving out the sodium and leaving the aluminum spongy and weak, fit only to be remelted. After remelting, however, it is purer and better in quality for its freedom from sodium.

Aluminum is unaffected by either concentrated sulphuric or nitric acids. Hydrochloric acid is its natural solvent; and when either sulphuric or nitric acid is contaminated with any hydrochloric acid, even though in very small proportions, it rapidly corrodes aluminum, the hydrochloric acid forming chloride of aluminum, which, in turn, is converted into sulphate or nitrate, the hydrochloric acid being set free in a nascent state to attack the aluminum again. In this way the corroding chloride acts as a carrier for the sulphuric or nitric acid.

Aluminum is not acted upon by sulphuretted hydrogen, carbonic oxide or carbonic acid gas, or by sulphurous acid or other sulphur vapors. These facts add much to the value of aluminum for many purposes where the tarnishing of silver is a serious inconvenience.

It is also practically unaffected by common salt, either wet or dry, or by sea-water, or by weak solutions of salt in acetic acid.

A piece of aluminum $3\frac{3}{8}$ inches by $1\frac{1}{2}$ inches by $\frac{7}{32}$ inches was immersed for twenty-three days in a 3 per cent. solution of common table-salt at a temperature of 80 degrees Fahrenheit. The strip lost 0.025 gramme in weight; 4.27 square inches were exposed; this would give a loss by corrosion per square inch per week of 0.00178 gramme. A similar strip was immersed for a similar length of time, under like conditions, in a solution of 3 per cent. of common salt with 2 per cent. of No. 8 acetic acid. The action in this case was not

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confined to that portion immersed in the liquid, but a crust of basic salts was also formed on the portion of the plate above the liquid. The strip was frequently reversed end for end in order to equalize the action. This piece lost 0.165 gramme upon an area of 6.4 inches, and the loss was at the rate of 0.00785 gramme per square inch per week.

These solutions were chosen as fairly representing extreme conditions to which aluminum would be subjected in domestic culinary operations. This corrosion is very slight and of no practical consequence, being much less than tin plate or silver plate would suffer under similar circumstances.

Solutions of the caustic alkalies readily attack aluminum.

In ammonia, aluminum is turned gray in color, but does not lose weight or strength. Chlorine, bromine, iodine and fluorine attack aluminum and corrode it.

Aluminum is not acted upon by organic secretions, such as sweat, saliva or the like; and the metal is coming to be used by dentists in the metal plates upon which to back false teeth, as well as by surgeons in tracheotomy-tubes, suture-wires, etc. For these purposes, aluminum should be free from iron, as it is found that when iron is present, the metal is acted upon by the saliva.

Mechanical Properties.

Experiments with a drop-testing-machine of the Pittsburgh Testing Laboratory, in which an accurately-ground, hardened steel point of 60 degrees, centrally-fixed in a weight of 32 pounds, dropped in guides a distance of 24 inches, resulted as follows:

Table V.

Drop-Tests.

Description of material.	Rolled or cast.	Depth and diameter of punch-mark.
		Inches.
1. Aluminum ¼-inch thick.....	Rolled.	.140 x .28
2. " " " "	"	.144 x .30
3. " " "	"	.143 x .30
4. " ½-inch thick.....	Cast.	.176 x .32
5. " " "	"	.177 x .32
6. " 17/32-inch thick....	"	.145 x .29
7. " " "	"	.144 x .29
8. " ⅜-inch thick.....	"	.177 x .32
9. Brass 7/32-inch thick....	"	.145 x .26
10. " " "	"	.142 x .26
11. " ¼-inch thick.....	Rolled.	.129 x .24
12. " " "	"	.129 x .24
13. Copper 7/32-inch thick....	"	.111 x .24
14. Zinc ¼-inch thick.....	"	.118 x .24
15. " " "	"	.111 x .24

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These results show only partially the difference in the surface-hardness of aluminum due to cold-working; castings of aluminum, made a little larger than the finished object desired, and drop-forged in dies, will be increased in hardness and rigidity, and thus adapted for tools and for many bearings, such as occur in the parts of surveying instruments, where great weight does not have to be sustained.

Rods, wire, and sheets of aluminum, similarly hardened by cold-rolling or cold-drawing, will have a hardness nearly equal to that of brass, and a good deal of elasticity and spring. The Eureka Tempered-Copper Company, Northeast, Pa., has succeeded in hardening castings of aluminum in the same way as it hardens copper. As already observed, the most careful and delicate tests have failed to show the slightest hardness given to aluminum by the process of "chilling," as practiced with cast-iron. The surfaces of the metal poured against the cold, hard, metal surfaces seem as soft as any other portions of the same aluminum.

Pure aluminum is one of the most ductile and malleable of metals. It can be drawn into the finest of wire, rolled to sheets of five ten-thousandths of an inch thickness, or hammered into foil as thin as any beaten gold-leaf. The relative malleability of aluminum, as shown by its capacity to beat out well into thin foil, is one of the best tests of the purity of the metal. The less silicon and iron in the metal, the better and more readily it works up under the beater's hammer. Metal of more than 1 per cent. of impurity will not beat out well into foil.

Pure aluminum is absolutely non-paramagnetic; the same remark practically applies to aluminum containing 0.05 per cent. of iron.

According to experiments made by Prof. Frank Very, of the Allegheny Observatory, a wire of the "average lot of 98.52 per cent. aluminum," suspended by a nearly torsionless cord, exhibited no appreciable polarity. An ingot of aluminum, containing 1.5 per cent. of iron, showed a very faint polarity; with 2 per cent. of iron, the polarity was distinct and very decidedly marked.

All the commercially pure aluminum sold by the Pittsburgh Reduction Company contains less than 0.20 per cent. of iron, much of it running as low as 0.04 per cent. of iron. For this reason, it will answer very well for compass-boxes, or cases for electrical apparatus, where non-magnetic properties are desirable. There are very few commercial metals, not chemically pure, containing as little iron as does the aluminum made by the Pittsburgh Reduction Company. Certainly, nearly all of the brasses, bronzes, or German-silver alloys contain a larger percentage of iron.

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The tensile, crushing, and transverse tests of aluminum vary very considerably with the varying conditions of hardness, due to cold-working. They are affected also by the amount of work that has been put upon the metal, the character of the section, etc., as will be shown by the following tables of tests, selected from very many, as averages of what the metal will stand:

TABLE VI.

Tensile and Compression-Tests of Aluminum in Various Shapes.

Size, shape and condition of aluminum tested.	Elastic limit, pounds per square inch in tension.	Ultimate strength, pounds per square inch in tension.	Percentage of reduction of area in tension.	Estimated safe unit strain per square inch in tension. Pounds.	Original size of specimens used in compression.	After failure, size of specimen used in compression.	Elastic limit per square inch in compression.	Ultimate strength per square inch in compression.
Metal of from 97 per cent. to 98.50 per cent. aluminum.								
Ordinary castings of over ½ inch sectional area as cast	6,182	15,640	8.66	3,000	1 in. high × ½ in. diam.	.985 × .503	3,565	12,730
Same annealed	6,600	17,100	15.00	3,500	" "	.979 × .506	2,057	10,185
Same lot of castings hardened by drop forging	10,000	24,000	18.00	3,000	" "	.965 × .508	4,583	15,275
Aluminum rolled bars left hard	16,000	32,500	42.00	6,000	1½" high × ¾" diam.	1.459 × .760	2,263	9,064
Same annealed	13,870	25,000	49.11	6,000	" "	1.468 × .756	2,043	6,809
Aluminum hammered bars left hard	14,630	24,120	35.14	5,500	" "	2.91 × .152	5,659	11,320
Same annealed	14,000	22,000	43.00	5,500	" "	2.91 × .151	5,094	11,320
Aluminum wire from ⅛ in. to 1.16 in. diameter as drawn	12,000	24,570	58.46	4,500				
Same annealed	10,870	22,830	65.57	4,500				
Aluminum wire No. 14 to 20 B. & S. gauge as drawn	32,000	54,000	68.00	12,000				
Aluminum wire No. 20 to 40 B. & S. gauge as drawn	26,000	49,000	54.00	10,000				
Same annealed	21,000	33,000	62.00	7,000				
Aluminum plates to ¼ in. thick as rolled	9,300	22,740	12.40	4,000				
Same annealed	8,500	17,740	13.73	4,000				
Aluminum sheets No. 8 to No. 20 B. & S. gauge as rolled	12,500	25,000	37.50	4,500				
Same annealed	10,700	21,070	39.00	4,500				
Aluminum sheets No. 20 to No. 30 B. & S. gauge as rolled	13,500	27,500	40.00	4,500				
Same annealed	11,000	23,000	43.50	4,500				
Metal of 94 per cent. aluminum, 1.50 per cent. iron, and 4.50 per cent. silicon.								
Sheets of ⅛ in. thick	12,000	29,650	11.00	4,500	1" high × ½" diam.	.965— .508	4,800	15,275
Castings ¼ in. x 1 in.	10,500	25,550	9.50	4,500				

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TABLE VII.

Compression-Tests of Cylinders of the "Average Lot, 98.52 per cent., Aluminum."

Beam Load.	Height x Diameter.					
	Q	O	P	R	N	M
0,000...	3. x 1.499	3. x 1.500	3. x 1.500	3. x 1.486	3. x 1.500	3. x 1.498
6,000...	3. x 1.499	3. x 1.500	3. x 1.500	3. x 1.486	3. x 1.500	3. x 1.498
8,000...	3. x 1.499	3. x 1.500	3. x 1.500	2.98 x 1.489	3. x 1.500	2.99 x 1.500
9,000...	2.99 x 1.501	2.995 x 1.5005	2.99 x 1.502	2.97 x 1.490	3. x 1.500	2.98 x 1.502
		Full		Full		
10,000...	2.98 x 1.503	2.99 x 1.502	2.985 x 1.503	2.96 x 1.491	2.99 x 1.502	2.97 x 1.504
12,000...	2.97 x 1.505	2.98 x 1.504	2.98 x 1.505	2.96 x 1.495	2.98 x 1.505	2.96 x 1.506
14,000...	2.96 x 1.507	2.97 x 1.505	2.97 x 1.507	2.95 x 1.497	2.96 x 1.507	2.91 x 1.520
		Full				
16,000...	2.96 x 1.508	2.96 x 1.504	2.95 x 1.510	2.94 x 1.501	2.95 x 1.509	2.84 x 1.540
18,000...	2.95 x 1.512	.....	2.94 x 1.514	.....	2.94 x 1.514	2.75 x 1.568
20,000...	2.93 x 1.516	2.94 x 1.514	2.92 x 1.519	2.90 x 1.510	2.92 x 1.519	2.64 x 1.602
25,000...	2.86 x 1.532	2.98 x 1.530	2.86 x 1.532	2.84 x 1.528	2.86 x 1.533	2.50 x 1.655
30,000...	2.78 x 1.559	2.81 x 1.555	2.77 x 1.560	2.75 x 1.554	2.79 x 1.552	2.22 x 1.766
100,000...	....	2.36 x 1.295	.....	....	.....	1.975 x 1.910
....	....	....	....	....	....	1.332 x 2.310
	Q	O	P	R	N	M

TABLE VIII.

Compression-Tests of Aluminum Cylinders of the "Average Lot."

Beam Load.	Height x Diameter.		
000.....	1.496 x .750	1.500 x .750	1.494 x .749
700.....	1.496 x .750	1.500 x .750	1.494 x .749
900.....	.....	1.4995 x .750	1.493 x .749
1,000.....	1.495 x .751	1.499 x .7505	1.491 x .749
1,200.....	1.493 x .751	1.499 x .7505	1.490 x .750
1,400.....	1.491 x .751	1.498 x .751	1.489 x .750
1,600.....	1.489 x .752	1.498 x .751	1.487 x .751
2,000.....	1.485 x .753	1.495 x .752	1.485 x .751
2,500.....	1.480 x .754	1.488 x .753	1.482 x .752
3,000.....	1.474 x .755	1.487 x .753	1.468 x .756
4,000.....	1.459 x .760	1.472 x .758	1.460 x .758
5,000.....	1.433 x .768	1.445 x .766	1.440 x .762
40,000.....	To shape J	....	....
67,000.....		To shape K	L.

NOTE.—The letters in these tables were stamped upon the samples for identification. "To shape J" or "K" in the above table means that at the corresponding pressure the cylinders were forced to the shapes of the samples so stamped.

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TABLE IX.

Compression-Tests of Cylinders of "Average Lot, 98.52 Per cent. Aluminum."

Beam load.	Height x Diameter.								
	A	B	C	D	E	F	G	H	I
000	1. x .500	.998 x .500	.990 x .500	1.00 x .500	.988 x .500	1.00 x .500	.995 x .498	1.001 x .500	1.00 x .499
400	.999 x .501	.998 x .500	.....	.....	.....	.....	.....	.....	.....
500	.998 x .501	.998 x .500	.....	1.00 x .500	.985 x .500	1.00 x .500	.995 x .498	1.001 x .500	1.00 x .499
600	.9975 x .5015	.997 x .500	.....	.....	.....	.....	.....	.....	.....
700	.997 x .5015	.996 x .5005	.....	.999 x .500	.985 x .500	Lost.	.993 x .498	1.00 x .500	1.00 x .499
800	.995 x .502	.996 x .5005	.....	.....	.....	.....	.....	.....	.....
900	.995 x .502	.995 x .5005	.....	.998 x .500	.983 x .500	.997 x .500	.991 x .499	1.00 x .500	.999 x .500
1,000	.995 x .502	.994 x .501	.998 x .500	.997 x .500	.981 x .500	.996 x .501	.990 x .499	.999 x .500	.998 x .500
1,100	.993 x .502	.....	.....	.....	.....	.....	.....	.....	.....
1,200	.992 x .502	.992 x .501	.984 x .501	.996 x .500	.976 x .501	.995 x .501	.987 x .500	.9975 x .500	.997 x .500
1,300	.992 x .502	.....	.....	.....	.....	.....	.....	.....	.....
1,400	.....	.990 x .5015	.980 x .501	.991 x .501	.975 x .501	.993 x .502	.984 x .501	.995 x .501	.996 x .500
1,500	.990 x .505	.....	.....	.....	.....	.....	.....	.....	.....
1,600	.....	.986 x .503	.977 x .501	.990 x .502	.974 x .5015	.990 x .502	.984 x .501	.993 x .501	.995 x .500
1,700	.988 x .506	.....	.....	.....	.....	.....	.....	.....	.....
1,800	.....	.984 x .503	.974 x .501	.987 x .503	.969 x .503	.987 x .504	.982 x .501	.990 x .502	.994 x .500
2,000	.979 x .506	.982 x .504	.971 x .502	.985 x .503	.966 x .504	.986 x .501	.975 x .503	.987 x .503	.991 x .502
2,500	.966 x .510	.972 x .5065	.952 x .507	.967 x .508	.953 x .507	.975 x .507	.963 x .507	.980 x .505	.984 x .503
3,000	.960 x .512	.960 x .513	.948 x .510	.952 x .510	.937 x .513	.961 x .512	.947 x .512	.955 x .512	.965 x .508
3,500	.....	.....	.....	.....	.....	.....	.....	.....	.988 x .517
17,000	.357 x .863	.....	.....	.....	.....	.....	.....	.....	.....

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TABLE X.

Compression-Tests of Best-Quality Charcoal Cast-Iron Cylinders.

Beam Load.	Height x Diameter.		
0,000	3. x 1.500	1.500 x .750	1. x .500
5,000		1.500 x .750	
8,000		1.500 x .750	
10,000		1.500 x .750	
15,000		1.500 x .750	
17,000			.969 x .509
19,000			.961 x .512
20,000	3. x 1.500	1.498 x .750	
23,000			Fracture.
30,000	3. x 1.500		
50,000	3. x 1.500		
50,500		Fracture.	
65,000	2.9995 x 1.500		
105,000	2.998 x 1.507		
	C. 1.	C. 2.	C. 3.

TABLE XI.

Compression-Tests of 12% Carbon Rivet-Rod Bessemer Steel Cylinders.

Beam Load.	Height x Diameter.		
0,000	3. x 1.500	1.495 x .749	1. x .500
1,000			1. x .500
1,500			.999 x .500
2,500			.999 x .500
3,000			.999 x .500
4,000			.999 x .500
5,000			.999 x .500
5,500			.998 x .500
6,000			.997 x .500
6,500			.996 x .500
7,000			.995 x .500
7,500			.992 x .5005
8,000		1.495 x .749	.980 x .503
10,000		1.494 x .749	
12,000		1.494 x .749	
15,000		1.492 x .749	
18,000		1.486 x .749	
20,000		1.482 x .749	
25,000		1.420 x .770	
30,000	3. x 1.500		
35,000			.532 x .709
40,000	2.9995 x 1.500		
50,000	2.9995 x 1.500	1.094 x .891	
55,000	2.99 x 1.500		
60,000	2.98 x 1.501		
65,000	2.96 x 1.507		
75,000	2.94 x 1.514		
80,000		1.071 x .780	
87,000	2.91 x 1.520		
100,000	2.88 x 1.537		
	S. 1.	S. 2.	S. 3.

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TABLE XII.

Compression-Tests of the Average Lot of 98.52 Per Cent. Aluminum, with Comparative Tests of 5 Per Cent. Aluminum-Bronze, Best Gray Charcoal Iron and 12-Carbon Bessemer Rivet-steel Cylinders.

Marks.	Character of material.	Original Dimensions.			After Fracture, dimensions.		Elastic limit per sq. in.	Ultimate strength per sq. in.
		High.	Diam.	Area.	High.	Diam.		
A	Cast aluminum	1.00	.500	.19685	.979	.506	2,037	10,185
B	" "	.998	.500	.19635	.972	.5065	3,056	12,730
C	" "	.990	.500	.19635	.952	.507	5,092	12,730
D	" "	1.00	.500	.19635	.985	.503	3,565	12,730
E	" "	.985	.500	.19635	.958	.507	2,546	12,730
F	" "	1.00	.500	.19685	.975	.507	4,583	12,730
G	" "	.995	.498	.19478	.975	.503	3,594	10,270
H	" "	1.001	.500	.19635	.958	.512	3,565	15,275
I	" "	1.00	.499	.19556	.965	.508	4,583	15,275
J	" "	1.496	.750	.4418	1.459	.760	2,263	9,054
K	" "	1.500	.750	.4418	1.472	.758	2,037	9,054
L	" "	1.494	.749	.4406	1.468	.756	2,043	6,809
M	" "	3.00	1.498	1.7624	2.91	1.52	5,659	11,320
N	" "	3.00	1.500	1.7671	2.92	1.519	5,094	11,320
O	" "	3.00	1.500	1.7671	2.92	1.519	5,659	11,320
P	" "	3.00	1.500	1.7671	2.94	1.514	5,094	11,300
Q	" "	3.00	1.499	1.7648	2.95	1.512	5,100	10,200
R	" "	3.00	1.486	1.7341	2.90	1.510	4,614	11,534
1	5 pr ct. aluminum-bronze	1.02	74 x 74	.5476	0.56	.....	15,100	69,390
2	" " "	1.01	72 x 75	.540	0.58	.....	12,400	70,370
3 SSS.	" " "	1.02	71 x 75	.5325	0.55	.....	15,700	67,000
4 SSS.	" " "	1.02	74 x 72	.5328	0.53	.....	11,200	65,600
5 SSS.	" " "	1.02	73 x 73	.5329	0.54	.....	15,900	66,000
6 SSS.	" " "	1.01	73 x 75	.5475	0.54	.....	14,600	63,500
1 S.	" " "	.99	.97	.73898	0.44	.....	10,820	50,130
2 S.	" " "	1.01	1.13	1.002	0.45	.....	12,770	50,410
3 S.	" " "	1.00	1.13	1.002	0.44	.....	12,770	51,680
C. 1	Cast-iron	3.00	1.50	1.7671	.....	.....	36,783	166,000
C. 2	"	1.50	.75	.4418	.....	.....	45,270	114,309
C. 3	"	1.00	.50	.1963	.....	.....	86,600	117,170
S. 1	Steel	3.00	1.50	1.7671	.....	.....	22,640	.....
S. 2	"	1.495	.749	.4406	1.420	.770	22,700	56,740
S. 3	"	1.00	.500	.1963	.980	.503	7,641	40,750

NOTE.—In all these tables, where not otherwise stated, the weights are in pounds and the dimensions in inches.

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TABLE XIII.

Tensile Tests of "Average Lot of 98.52 Per Cent. Aluminum," Aluminum Bronze and Wrought-Iron Containing Aluminum.

Marks.	Character of material.	Original.		After fracture.		Reduction area per cent.	Elastic limit per sq. in.	Tensile strength per sq. in.
		Dimens'ns.	Area.	Dimens'ns.	Area.			
1	¼ in. thick aluminum sheet...	1.002 x .244	.2445	.965 x .215	.2075	15.13	.....	22,050
2	" " " casti'g	1.004 x .251	.2520	.982 x .247	.2426	3.73	.....	17,740
3	" " " sheet..	1.003 x .238	.2387	.968 x .216	.2091	12.4	.....	22,040
4	No. 13 B. & S. gauge, alumi'm	1.00 x .073	.0730	.990 x .068	.0673	7.81	.....	37,950
5	" " " "	1.00 x .072	.0720	.944 x .064	.0604	16.11	.....	30,420
.....	3" x 1" ingot alum. casting.....	2.91 x .992	2.887	2.82 x .935	2.637	8.66	6,132	15,640
.....	No. 8 B. & S. gauge, alum. sheet	3.01 x .125	.3763	2.66 x .072	.1915	49.11	6,218	17,410
.....	¼ in. r'd. Drawn wire rod.....	.242 diam.	.0460	.142	.01584	65.57	10,870	22,830
.....	" " " "	.242 "	.0460	.156	.01911	58.46	12,170	24,570
.....	No. 14 B. & S. gauge, alum. sheet	1.97 x .062	.1221	1.95 x .043	.08385	31.33	14,500	24,980
.....	" " " " "	1.975 x .063	.1244	1.95 x .043	.08385	32.6	14,630	24,120
.....	" " " " "	2.00 x .061	.1240	1.95 x .044	.0858	30.81	13,870	25,000
.....	" " " " "	2.015 x .060	.1209	1.98 x .046	.09108	24.67	12,570	23,990
.....	" " " " "	1.945 x .060	.1167	1.892 x .037	.0700	40.02	17,560	25,710
.....	No. 8 " " " "	1.915 x .122	.2336	1.882 x .079	.1487	36.34	18,620	26,540
.....	" " " " "	2.007 x .120	.2408	1.968 x .080	.1574	34.63	19,100	26,990
.....	" " " " "	1.985 x .122	.2422	1.944 x .079	.1536	36.58	16,510	26,630
1	Aluminum.....	.75 x .69	.5175	.....	.4680	9.5	5,980	14,760
2	"	.75 x .71	.5325	.....	.4757	10.6	6,000	18,460
3 888.	"	.76 x .70	.5320	.....	.4818	9.4	7,510	12,760
4 888.	"	.75 x .70	.5250	.....	.4784	8.8	6,850	12,830
5 888.	"	.75 x .70	.5250	.....	.4615	12.0	8,090	14,430
6 888.	"	.75 x .73	.5475	.....	.4422	19.0	7,430	16,430
1 S.	"	.97 diam.	.73898	.....	.69397	6.0	2,700	8,480
2 S.	Aluminum cast cylinders.....	1.34	1.4102	.....	1.327	5.8	2,090	8,730
3 S.	" " "	1.33	1.3892	.....	1.327	4.4	4,650	7,570
7	5 per cent. aluminum bronze.	.377 x .255	.09613	.....	.0651	32.2	31,410	71,250
8	" " "	.376 x .254	.0955	.....	.0651	31.7	41,250	69,520
9	" " "	.377 x .254	.09575	.....	.0651	32.0	20,880	70,600
10	" " "	.376 x .257	.09663	.....	.058	39.9	31,040	79,270
1.1	Sin. roll. iron muck bar 1" sq.	1.00 x 1.00	1.000	.795 x .790	.6281	37.19	27,800	47,000
1.2	" " " "	1.005 x 1.00	1.005	.780 x .777	.6061	39.69	28,060	47,270
1.3	" " " "	1.00 x 1.00	1.000	.775 x .771	.5975	40.25	28,650	47,150
2.1	Same single rolled iron muck bar, 3 layers, with 2 layers aluminum muck bar, 1" sq..	.997 x .999	.9960	.790 x .790	.6241	37.34	28,510	49,190
2.2	Same single rolled iron muck bar, 3 layers, with 2 layers aluminum muck bar 1" sq..	1.00 x .998	.9980	.795 x .775	.6161	38.26	28,650	48,890
2.3	Same single rolled iron muck bar, 3 layers, with 2 layers aluminum muck bar, 1" sq..	1.00 x 1.00	1.000	.785 x .785	.6162	38.38	29,070	49,450
3.1	All aluminum muck bar, 3 per cent. aluminum, 1" sq..	1.00 x .994	.9940	.880 x .868	.7638	23.17	28,604	47,990
3.2	All aluminum muck bar, 3 per cent. aluminium, 1" sq.	1.001 x 1.00	1.001	.870 x .875	.7613	23.95	28,770	48,100
3.3	All aluminum muck bar, 3 per cent. aluminum, 1" sq..	1.002 x 1.00	1.002	.875 x .870	.7613	24.02	28,940	48,650
.12-carb.	Bessemer rivet steel.....	¾" Dia. .765	.4593	.460	1.662	63.82	39,410	57,590
	" " " "	1⅝ " 1.369	1.469	.940	.6940	54.11	36,690	60,850
.....	.....	.....	.....	.....	.....	.....	.....	.....

Aluminum is in no sense a very rigid metal, as is shown by the following tests:

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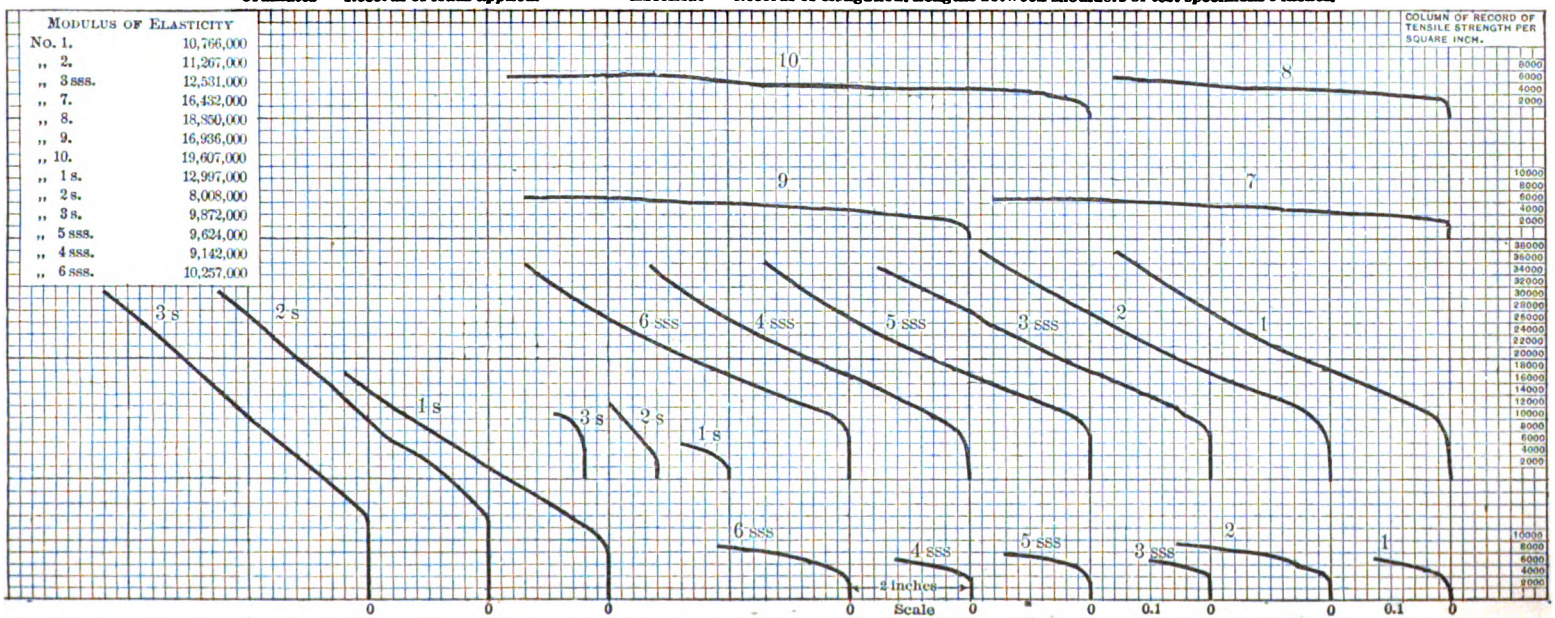
TABLE XIV.

Record of Elongations, Elastic Limits and Tensile Strengths of Specimens Aluminum and Aluminum Bronze, made upon a 100,000 pound new model Olsen Testing Machine with new automatic device for plotting the records given by the machine.

Figures on curves Refer to Specimen Numbers on attached tables.

Ordinates——Records of loads applied. Abscissae——Records of elongation. Lengths between shoulders of test specimens 8 inches.



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A 1-inch square bar of "the average lot 98.52 per cent. aluminum," supported on knife-edges 24 inches apart, loaded at the center with single load, gave the following results:

TABLE XV.

Transverse Tests of Aluminum.

Load. Pounds.	Deflection. Inches.	Permanent Set. Inches.
50	$\frac{2}{64}$	$\frac{2}{64}$
100	$\frac{5}{64}$	.....
150	$\frac{8}{64}$	$\frac{1\frac{6}{64}}{64}$
175	$\frac{10}{64}$	.....
200	$\frac{12}{64}$	$\frac{4\frac{0}{64}}{64}$
225	$\frac{14}{64}$	.....
250	$1\frac{1}{64}$	$1\frac{1\frac{6}{64}}{64}$
275	$1\frac{5\frac{0}{64}}{64}$	$1\frac{5\frac{0}{64}}{64}$
300	$2\frac{1}{64}$	$2\frac{2}{64}$
	not ruptured.	

Another lot of "the average 98.52 per cent. aluminum," in the shape of a $1\frac{1}{2}$-inch round bar, supported on knife-edges 12 inches apart, loaded at the center with single load, gave the following results:

TABLE XVI.

Transverse Tests.

Load. Pounds.	Deflection. Inches.	Permanent Set. Inches.
0	0	.....
100	0	.....
200	0	.....
300	0	.....
400	0	.....
500	0	.....
600	0	.....
700	0	.....
800	$\frac{1}{64}$	.....
900	$\frac{1}{64}$	.....
1000	$\frac{2}{64}$	0
1100	$\frac{2}{64}$	.....
1200	$\frac{2}{64}$	.....
1300	$\frac{3}{64}$	.....
1400	$\frac{3}{64}$	.....
1500	$\frac{4}{64}$	$\frac{1}{64}$
1600	$\frac{4}{64}$	.....
1700	$\frac{5}{64}$	.....
1800	$\frac{5}{64}$	.....
1900	$\frac{6}{64}$	.....
2000	$\frac{6}{64}$	$\frac{7}{64}$
2100	$\frac{7}{64}$	.....
2200	$\frac{7}{64}$	.....
2300	$\frac{8}{64}$	.....
2400	$\frac{8}{64}$	.....
2500	$\frac{9}{64}$	.....
2600	$\frac{9}{64}$	.....
2700	$\frac{10}{64}$	.....
2800	$\frac{10}{64}$	Broke.

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Neither of these tests seemed to follow the general formula for deflections, viz., $D = \frac{Pl^3}{48EI}$, when D = deflection; P = load; l = length; E = modulus of elasticity of the metal (taken for aluminum bars as 13,000,000); I = element of inertia of the action. Neither will any other assumed modulus of elasticity for aluminum give results agreeing with the deflection observed under these transverse tests.

The flow of aluminum under tensile tests is very local; the percentage of elongation reducing very rapidly as it is calculated in increasing lengths from the point of fracture.

From the results reached as averages we submit the following statement of what we believe will be found to be average tensile and compression-tests of commercially pure aluminum, of composition varying in each constituent as follows:

Aluminum, from 97 to 99 per cent.
Silicon, graphitic, from 0.10 to 1.00 per cent.
Silicon, combined, from 0.90 to 2.80 per cent.
Iron, from 0.04 to 0.20 per cent.

TABLE XVII.

Average Tensile and Compression-Tests of Commercial Aluminum.

		Pounds.
Elastic limit per square inch in tension,	Castings,	6,500
" " "	Sheet,	12,000
" " "	Wire,	16,000–30,000
" " "	Bars,	14,000
Ultimate strength per square inch in tension,	Castings,	15,000
" " "	Sheet,	24,000
" " "	Wire,	30,000–65,000
" " "	Bars,	28,000
		Per cent.
Percentage of reduction of area in tension,	Castings,	15
" " "	Sheet,	35
" " "	Wire,	60
" " "	Bars,	40
		Pounds.
Elastic limit per square inch under compression in cylinders, with length twice the diameter,		3,500
Ultimate strength per square inch under compression in cylinders, with length twice the diameter,		12,000

The modulus of elasticity of cast aluminum is about 11,000,000.
The modulus of elasticity of cold-drawn aluminum wire is about 19,000,000.
The modulus of elasticity of aluminum sheets and bars is about 13,000,000.

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Taking the tensile strength of aluminum in relation to its weight, it is as strong as steel of 80,000 pounds per square inch. Comparative results calculated in this way are tabulated below, as taken from Richards's *Aluminium*.

TABLE XVIII.

Tensile Strengths.

	Weight of 1 cubic foot in pounds.	Tensile strength per square inch.	Length of a bar able to support its weight in feet.
Cast-iron	444	16,500	5,351
Ordinary bronze	525	36,000	9,893
Wrought-iron	480	50,000	15,000
Hard structural steel	490	78,000	23,040
Aluminum	168	26,800	23,040

Mierzinski is quoted in Dingler's *Magazine* as giving the following results of tensile tests of aluminum wire:

TABLE XIX.

Tensile Tests by Mierzinski.

Diameter. Millimeters.	Tensile strength in grammes.			Tenacity. Kilos per square millimeter.	Pounds per square inch.
	1st trial.	2d trial.	Mean.		
0.225	661	653	657	12,975	18,360
0.205	524	506	515	12,255	7,427
0.160	307	311	309	12,700	18,059
0.145	246	252	249	11,845	16,844

We have, however, obtained much higher results in tensile strength per square inch, and exhibit with this paper aluminum wire of No. 2 B. & S. gauge which pulled 49,500 pounds per square inch.

Mr. Spilsbury, of the Trenton Iron Company, informs us that he has obtained, on aluminum wire of .061-inch diameter, 62,300 pounds per square inch tensile strength; the wire being drawn with one annealing from wire-rod of .160-inch diameter. The same wire made 47 bends to a right angle before breaking.

Aluminum, when pure, is a very sonorous metal; bars of it suspended by fine wire, when struck, give a fine clear bell-like sound. The proper shapes for bells have not yet been devised. Bells of ordinary shapes do not give as good sound as do ingots of the metal.

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Conductivity and Electro-Positiveness.

Mr. C. K. McGee, of the University of Michigan, has determined the electrical resistance of specimens of the "average lot 98.52 per cent. aluminum" metal, with the following results:

Table XX.

Electrical Resistances.

Specimen used.	Diameter in inches.	Resistance in "legal ohms" of a yard,	
		At 14°C [=57°F].	At 76°C [=170°F].
Unannealed aluminum wire	.0325	.05749	.07202
Annealed aluminum wire	.0325	.05484	.06928
Pure copper wire has the following resist'nce	.0325	.0315	.0387

In the electro-chemical series, aluminum is ordinarily placed near the positive end, being, under most circumstances, more positive than almost all the other metals, and only less positive than the metals of the alkalies and alkaline earths. That is, in most separations of aluminum by electrolysis it is charged with positive electricity, and separates at the negative pole.

This arrangement is only approximate, however. Under some circumstances aluminum is electro-positive to sodium, as it reduces sodium when treated with its oxide or carbonate; again, it is electro-negative to iron, as iron reduces aluminum from its sulphide.

The attraction of aluminum for oxygen is exceeded by that of very few elements. If this attraction be measured by the amount of heat developed in the combination of aluminum and oxygen, it is about three times that of carbon for oxygen. That is to say, one equivalent of carbon uniting with two of oxygen gives 96,000 units, while two of aluminum with three of oxygen give 388,000 units of heat.

In its behavior with most chemical reagents, at ordinary temperatures, aluminum resembles platinum and gold. It is like carbon and silicon, which are highly electro-positive at high temperatures, but electro-negative at low. Under some circumstances, as in nitric acid, aluminum is as electro-negative as platinum or carbon.

It is suggested that at low temperatures the atoms of aluminum are combined with each other so as to render the metal inert. If its

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full chemical affinities were exhibited at low temperatures, it would be as easily oxidized by acid and water as metallic sodium.

Action of Impurities.

As observed under the head of Purity of Aluminum, silicon and iron form the bulk of the impurities; and these two elements will be treated first.

Silicon hardens aluminum considerably, increasing its tensile strength without materially decreasing its ductility; it, however, very materially decreases its malleability, and takes away its capacity of taking a fine polish, and, much more, prevents its retaining whatever polish it has received. Silicon in aluminium oxidizes by action of the atmosphere or of moisture, and if present in proportions of over 3 per cent. very soon coats the metal with a dull gray and unsightly tarnish.

For some purposes, where a harder surface is required than is given by pure aluminum, where advantage would be taken of the lightness of the metal, and where the surface can be lacquered or otherwise coated to protect it from oxidation, the alloy, say 6 to 8 per cent. of silicon, which can be readily made, can be used with advantage; but in all ordinary work to which aluminum is put, the more nearly free the metal from silicon the better. These remarks apply to the ordinary ways in which we find silicon, a large proportion of which exists in the graphitoidal form. Whether the influence of the amorphous silicon, could it be placed in the metal alone, free of graphitoidal silicon, would give the advantageous hardness without the tarnishing qualities, is an interesting question not yet answered.

Iron in small percentages, as an impurity of aluminum, hardens it, but, of course, adds to its specific gravity and renders it magnetic. It also decreases the malleability of the metal, and, like silicon, detracts from the capacity of the metal to take a fine polish or to retain whatever polish it is at first susceptible of, although the alloy does not tarnish as rapidly as does the siliconized metal. For some purposes, where a harder and stouter alloy is wanted, a proportion of from 6 to 10 parts of iron works advantageously.

W. J. Keep has pointed out the curious fact that the alloy of 50 per cent. iron and 50 per cent. aluminum, though melted together and seemingly forming a true alloy in the pot and in the metal as first cast, seems entirely to lose its power of cohesion and crumbles

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down in a little while to a powder, each grain of which seems to contain equal parts of aluminum and iron.

Copper sometimes becomes an accidental impurity of commercially pure aluminum, in proportions up to perhaps one-half of one per cent.; in such small percentages its influence is hardly noticeable in any of the properties of the metal, so far as has come to our notice.

In larger proportions, up to 10 per cent., copper can readily and advantageously be alloyed with aluminum, especially for castings where hardness of surface is required. It takes away the peculiar polish of aluminum, and the fine gloss and peculiar color of cast aluminum; at the same time it adds hardness and decreases the shrinkage of castings, and for many purposes can be used advantageously. It has the disadvantage of adding materially to the specific gravity of the metal.

Carbon only unites with aluminum under very high and continued heat, and then only in proportions not exceeding 3 per cent. Whenever it is found associated with aluminum the metal is brittle, porous and friable.

Both Deville and Fremy say that, under ordinary temperatures, aluminum and sulphur do not unite or act on each other; indeed, Deville says aluminum may be heated in a glass tube to a red heat in vapor of sulphur without altering the metal. However, at very high temperatures, aluminum and sulphur do combine to form a sulphide, of the composition Al^2S^3. Ordinary aluminum of commerce is entirely free from sulphur.

Lead is found as an accidental impurity of aluminum in proportions up to one-quarter of 1 per cent. In these small proportions, so far as yet determined, it has no appreciable action upon the properties of aluminum. In larger proportions, lead does not alloy with aluminum, and no homogeneous alloy, or even mixture of the two metals, can be easily obtained.

Antimony does not unite with aluminum to form any homogeneous alloy.

Chromium unites with aluminum readily, hardening it and adding to its tensile strength. An addition of 2 to 3 per cent. of chromium to aluminum improves it for many purposes, although, as in the case of all other impurities, it decreases the malleability of the aluminum.

Tungsten unites with aluminum, hardening the metal, but not giving any very useful alloys.

Platinum unites with aluminum readily; it, however, does not seem to give any advantageous alloys; they are brittle and unsound.

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As to *Tiers Argent*, or aluminum and silver, Tissier says that silver seems to be the metal most useful to improve aluminum, 5 per cent. of silver giving to aluminum increased elasticity and hardness without injuring the malleability of the metal. The alloy is susceptible of a fine polish. The silver lowers the melting-point of the aluminum considerably. We predict that alloys of silver and aluminum will have a large use in the arts in the future.

No very valuable alloys of tin and aluminum have been discovered. Tin added to aluminum makes it more brittle, and does not seem to give any valuable properties in return. However, small proportions, up to 2 per cent., of aluminum added to tin render the tin harder and more elastic without decreasing materially its malleability.

Cadmium unites readily with aluminum, giving fusible alloys which are quite malleable; it, however, does not seem to impart strength, but rather weakness to the aluminum.

Bismuth unites with aluminum to form brittle, but very fusible, alloys.

Nickel, in any large proportions, unites with aluminum to form brittle alloys; in small percentages, up to 3 per cent. of nickel, its influence is to harden aluminum without seriously decreasing the malleability or ductility of the metal.

An alloy of 70 per cent. copper, 23 per cent. nickel, and 7 per cent. aluminum, has a fine yellow color, and takes a high polish. The addition of a small percentage of phosphorus seems to strengthen the alloy.

Zinc forms alloys readily with aluminum, which are very brittle and highly crystalline. The best solder yet obtained for aluminum is this alloy of zinc with it, using Venetian turpentine as a flux. Unfortunately it will not flow well, nor are the soldered surfaces capable of withstanding very hard usage.

The attraction between aluminum and phosphorus does not seem to be strong. It is difficult to cause them to unite, and small proportions of phosphorus do not apparently increase, when united with aluminum, its ductility, strength or toughness. Yet strength and toughness are increased when phosphorus is added to bronze, brass, or even aluminum-bronze *alloys*.

Arsenic does not readily unite with aluminum, and no valuable alloys of these two elements have been discovered. Aluminum, as manufactured, is never contaminated with arsenic.

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Practical Hints.

Dipping and Pickling.—Remove the dirt and grease from the plates by dipping in benzine. To whiten the metal, leaving on the surface a beautiful white mat, the sheet should be first dipped in a strong solution of caustic potash. It should then be dipped in a mixture of concentrated acids, two parts nitric acid to one of sulphuric acid; then in a solution of undiluted nitric acid; then in a mixture of vinegar and water, equal parts; then washed thoroughly in water, and dried as usual in hot sawdust.

To Polish.—Use a fine polishing composition or rouge, or tripoli, and a sheep-skin or chamois-skin buff; although it is often polished with an ordinary rag-buff.

For fine work, to polish aluminum, use a mixture of equal parts, by weight, of olive oil and rum, made into an emulsion by being well shaken together in a bottle. The polishing stone is dipped in this liquid and the metal is polished, without using, however, too much pressure.

Aluminum may be easily ground by using olive oil and pumice.

The surface of aluminum treated with a varnish of four parts oil of turpentine to one of stearic acid, or with a mixture of olive oil and rum shaken into an emulsion, allows an engraving tool to work on aluminum as on pure copper.

For Burnishing.—Use a blood-stone or steel burnisher. For hand-burnishing, use either kerosene oil or a solution composed of two tablespoonfuls of ground borax dissolved in about a quart of hot water, with a few drops of ammonia added.

For Lathe-Work.—The burnisher should wear upon the fingers of his left hand a piece of canton flannel, keeping it soaked with kerosene, and bringing it in contact with the metal, supplying a constant lubricant.

Very fine effects can be produced by first burnishing or polishing the metal and then stamping it in polished dies, showing unpolished figures in relief.

Scratch-Brushing.—Polish or burnish the surface, and then use a fine steel scratch-brush. A very fine finish is attained by rubbing with ground pumice-stone and water.

In spinning aluminum, plenty of oil should be used to prevent the clogging of the tool and to make it cut smooth in the turning, and to assist in the spinning.

To Solder the Metal.—Soldering the metal in large surfaces has not been successfully accomplished up to the present. Small sur-

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faces of the metal can be readily soldered by the use of pure zinc and Venetian turpentine. Place the solder upon the metal, with the Venetian turpentine, and heat gently with a blow-pipe until the solder is melted. It will then be found to have fixed itself firmly to the aluminum. The trouble with this, as with other solders, is, that it will not flow on the metal. Therefore, large surfaces are not easily soldered.

In cold-rolling aluminum, upon rolls designed for cold-rolling hard crucible steel, it has been found possible to reduce aluminum through the same sections as hard steel; the aluminum required, on the average, five annealings, where the steel required three to withstand the same work satisfactorily.

Alloys of Aluminum and Copper.

Ten per cent. aluminum, with 90 per cent. copper (called 10 per cent. aluminum bronze), rolled into plates, has an elastic limit of 70,000 to 80,000 pounds per square inch, a tensile strength of from 100,000 to 120,000 pounds per square inch, a reduction of area of from 20 to 40 per cent., with an elongation of from 5 to 10 per cent. in 8 inches. The metal is a beautiful yellow color, and is susceptible of taking a fine polish. One great advantage of the metal over any other varieties of bronze or brass is its comparative freedom from corrosion through the action of air, either moist or dry, or water, upon it. Its specific gravity, in castings, is about 7.84, and in rolled sheets about 7.89. Its modulus of elasticity is about 18,000,000 pounds. In castings it has a tensile strength of between 70,000 and 80,000 pounds per square inch, with a reduction of area of about 20 per cent.

Compression-tests upon 10 per cent. aluminum-bronze, $\frac{3}{4}$ inch in diameter, and 2 inches long, gave an ultimate compressive strength of 160,000 pounds per square inch, the specimens being shortened by $\frac{1}{4}$ inch. A similar piece of 5 per cent. bronze was shortened to $1\frac{1}{2}$ inches, and gave an ultimate compressive strength of 153,000 pounds per square inch.

Five per cent. aluminum-bronze in tension has an elastic limit of about 50,000 pounds per square inch; a tensile strength of about 70,000 pounds per square inch, a reduction of area of from 30 to 50 per cent. Its specific gravity is from 8.20 to 8.30. Two and one-half per cent. aluminum-bronze has a specific gravity of 8.6.

The melting point of 10 per cent. bronze is about 1700° Fahrenheit—a little higher than that of ordinary brass. The metal shrinks

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a little less than $\frac{1}{8}$ inch to the foot, or a little less than ordinary brass. It solidifies very rapidly from the molten condition, and it is necessary to pour very quickly. The feed-gates should be made large enough to prevent the metal freezing. Hot-baked sand moulds should be used for casting. Precautions should be taken also to prevent oxidation of the metal, for without it, the oxide is carried into the metal, which prevents its rolling into sound sheets. It is well, also, to bottom-pour the metal into the mould—that is, to cast the metal into a hot ladle having a nozzle in the bottom in direct connection with the gate of the casting, allowing the metal to settle, so that the oxide and dross shall come to the surface, in this way preventing its entering into the casting below. The surface of the molten bath should be kept covered with powdered charcoal. It is also advantageous to keep the bath covered with a flux in some cases, although the disadvantage of this is, that the flux is apt to cut the sides of the pot and add silicon to the metal. It is well to tap the metal in an inert atmosphere (casting in a cloud of smoke or the like) to prevent the oxidation from the air in the mould attacking the metal.

Aluminum-bronze is an extremely dense, close metal. It can be worked at a bright-red heat as easily as can wrought-iron. In this respect it differs from all other forms of bronze, which are red-short at a red-heat. The fact that aluminum-bronze is malleable at a red heat, and stands this temperature without change, makes it especially adaptable for blast-furnace tuyeres. The metal can be hardened to a considerable extent by working without annealing.

To anneal aluminum-bronze, heat to a dull red heat and permit it to cool gradually.

The alloy of aluminum and copper does not volatilize at any ordinary temperature used in fusing it, and consequently it can be frequently remelted without any appreciable change in the chemical constituents of the metal. This has great advantages in the economic use of the metal, as the scrap in castings or rolling can be readily remelted into ingots of the same quality.

Aluminum-bronze can be brazed as well as any other metal, using as a solder: Zinc, 50 per cent.; copper, 50 per cent.; using $\frac{1}{4}$ of the solder and $\frac{3}{4}$ of borax and cryolite in equal parts.

With pure aluminum, as now manufactured by the Pittsburgh Reduction Company, very pure aluminum-bronze alloys can readily be made, the impurities in the aluminum being reduced to one-tenth their amount on being diluted with pure electrolytic Lake copper.

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The following are some of the analyses of aluminum-bronzes lately made by the Pittsburgh Reduction Company and by the Scovill Manufacturing Company:

TABLE XXI.

Analyses of Aluminum-Bronze.

Kind of Alloy.	Per ct. aluminum.	Per ct copper.	Per ct. graphitoidal silicon.	Per ct. non-graphitoidal silicon.	Per ct. iron.	Specific gravity.
10 per cent. bronze casting made December 5th, 1889..	9.20	90.00	0.117	0.370	0.077	7.69
5 per cent. bronze casting made December 5th, 1889	4.70	94.84	0.080	0.320	0.060	8.25
2½ per cent. bronze casting made December 5th, 1889	2.35	97.29	0.050	0.260	0.050	8.61
10 per cent. bronze casting made at Neuhausen	6.32	91.98	0.090	1.090	0.480	8.01

ALLOYS OF ALUMINUM AND IRON.

Aluminum in Wrought-Iron.—The influence of aluminum in making wrought-iron fluid has been taken advantage of in the well-known Mitis process of making castings of wrought-iron. Aluminum furnished by the Pittsburgh Reduction Company has been found to be very advantageous, and is largely used, in the manufacture of Mitis metal. Aluminum will also increase the tensile strength of wrought-iron, and improve the fibre, if added, either as pure metal or in the form of ferro-aluminum, to the molten bath just before the metal comes to nature in the puddling furnace.

Aluminum in Cast-Iron.—The influence of aluminum in cast-iron is to turn the combined carbon to its graphitoidal state—that is, to make the white iron gray, and also to close the texture of the metal (W. J. Keep). It is also reported to make the metal ordinarily more fluid, and susceptible of taking a better polish and retaining it free from oxidation. Aluminum will also increase the tensile strength of many grades of cast-iron, and aids in obtaining sound castings free from blow-holes.

It has been used in preparations from one-tenth of 1 per cent. to 2 per cent., with good results, with various grades of iron.

Aluminum in Steel.—The influence of aluminum in steel of high carbon is to turn the combined carbon into graphite, and destroy the hardening action of the carbon in tool-steel. Aluminum in this

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sense softens steel. In structural steel, of .20 per cent. carbon, a small amount of aluminum, up to 1 per cent., increases the tensile strength without decreasing the ductility to any great degree. By its aid, a higher tensile strength can be obtained in thick sections of steel which have been subjected to but little work than can be otherwise obtained. Although aluminum, with considerable quantities of graphitoidal silicon, has been added to steel, no graphitoidal silicon has been found in the steel afterwards, it being all found in the amorphous or combined state in the resulting steel. It is claimed that the influence of aluminum is to lower the melting-point of steel, and in this way make it more fluid, and also to make the ingots of steel more solid and more free from blow-holes. It has been used in proportions of from one-tenth of 1 per cent. up to 3 per cent. of aluminum in the steel.

TABLE XXII.

Tensile Tests of Iron and Steel containing Aluminum.

CHARACTER OF MATERIAL TESTED.	Elastic limit per sq. in.	Tensile strength per sq. in.	Per ct. elongation in 8".	Per ct. reduction of area.	Character of fracture.	Analysis.				
						Per ct. aluminum.	Per ct. carbon.	Per ct. manganese.	Per ct. silicon.	Per ct. phosphorus.
All iron muck bar rolled, Union Mills of C., P. & Co., Limited	27,300	46,500	28.00	36.08	Fibrous.					
Three parts all iron with two parts aluminum muck	26,800	49,000	29.25	40.32	"					
All aluminum muck bar	29,500	55,300	14.00	22.56	"					
Four parts all iron with one part aluminum muck	28,510	49,190	29.50	37.34	"					
Open-hearth steel, with 1-10 of 1 per cent. aluminum, ¾ in. thick, plates	40,250	58,590	30.10	63.82	Silky cupped.	0.04	0.12	0.41	0.02	0.05
Open-hearth steel, with 1-10 of 1 per cent. aluminum, 1 in. thick, plates	47,950	68,360	25.30	52.10	Silky.	0.07	0.16	0.48	0.03	0.03
Open-hearth steel, with 1-10 of 1 per cent. aluminum, 2 in. thick, bar steel	46,260	63,700	26.10	47.90	Silky.	0.08	0.16	0.39	0.02	0.05
Open-hearth steel, with 1-10 of 1 per cent. aluminum, 2 in. thick, bar steel	45,980	67,560	29.2	54.80	Silky.	0.06	0.15	0.55	0.04	0.05

METHODS OF ANALYSIS OF ALUMINUM AND ITS ALLOYS.

As most of the properties of aluminum are materially affected by the purity of the metal, and as the literature of methods of analysis of aluminum metals and its alloys, is very scanty, we have considered it appropriate to append to this paper the methods used by the Pittsburgh Testing Laboratory, chemists for the Pittsburgh Reduction Company, in their various analyses referred to, which have been largely used as the basis of information of the conclusions drawn in this paper.

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SCHEME FOR ANALYSIS OF METALLIC ALUMINUM.

Iron.—Dissolve from 1 to 5 grammes of the metal in hydrochloric acid, and reduce with stannic chloride, taking up the excess of stannic chloride with bichloride of mercury. Titrate with a standard solution of bichromate of potash.

Silicon.—Dissolve 1 gramme of the metal in aqua-regia composed of equal parts of nitric and hydrochloric acids; evaporate to dryness; take into solution with 5 cubic centimeters of strong hydrochloric acid; add water, boil and filter off the gray residue; wash with dilute hydrochloric acid; ignite and weigh as silicon, plus silica; fuse with carbonate of soda; dissolve in dilute muriatic acid; evaporate to dryness; redissolve in dilute hydrochloric acid and filter off total silica. The difference between the weight of the silicon and the silica being the oxygen, which is united with the graphitoidal silicon, from which the amount of graphitoidal silicon can be calculated.

Copper and Lead.—Dilute the filtrate from the silica till free hydrochloric acid is only about 1 per cent. of the solution. Pass sulphuretted hydrogen through the hot solution, and filter off sulphides of copper and lead. Dissolve in 1.2 nitric acid and filter off sulphur. Add 5 c.c. sulphuric acid, and evaporate to sulphur fumes. Dilute, and, after waiting some time, filter from sulphate of lead into a platinum dish. The lead sulphate should be weighed, and lead calculated from it. Nearly neutralize with ammonia, leaving about 2 drops of sulphuric acid (dilute), and precipitate copper by battery.

Aluminum.—Receive filtrate from sulphides of copper and lead in 500 c.c. flask, and make up to the 500 c.c. mark with water. Take 50 c.c. of the solution (equal to $\frac{1}{10}$ gramme of the metal), boil off the free sulphuretted hydrogen, oxidize the iron in the solution with a little bromine water, and add sufficient ammonia to precipitate the sesquioxide of iron and alumina, which, after careful washing, ignite and weigh. Subtract the sesquioxide of iron calculated from the iron found by titration and the weight of any phosphoric acid which may be present, and calculate the remainder as Al^2O^3, from which the percentage of aluminum can be readily reckoned. (We have not found more than a trace of phosphorus in any of the samples of aluminum manufactured by the Pittsburgh Reduction Company.)

Calcium.—A trace may be found in the filtrates from iron and alumina, and is separated in the usual way. The filtrates from the

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calcium can be readily examined for magnesium. (We do not find any magnesium present in the metal manufactured by the Pittsburgh Reduction Company.)

Alkalies.—Dissolve 1 gramme in hydrochloric acid and evaporate in a platinum dish to separate the silica, and proceed as usual until the sesquioxides of alumina and iron are precipitated with ammonia. Dissolve this, after washing, in nitric acid, in platinum, and evaporate to dryness on the water-bath. Finally, heat over a small flame until the nitrate of alumina is all decomposed. This condition is indicated when the residue is white and friable, and no acid fumes can be detected by ammonia. Treat residue with hot water and get alkali nitrates in solution. This, together with the filtrate from alumina, contains all the alkali present, which should be evaporated, and after separating contained portions of lime and magnesia weighed as chlorides. (No sodium or potassium is found in the metal manufactured by the Pittsburgh Reduction Company.)

Methods of Analysis of Aluminum in Iron and Steel.

The accurate determination of aluminum when present in minute proportions, as less than .10 per cent., in iron or steel, has been a very difficult problem, requiring very skilled work to obtain accurate results. The method given by Mr. Andrew A. Blair, in his work upon the *Chemical Analysis of Iron*, is undoubtedly a correct one, and is to be recommended. It is, however, tedious, and requires several days to make the determination. The method of Mr. John E. Stead for the determination of minute quantities of aluminum in iron and steel has given very satisfactory results and is as follows:

Weigh off 11 or 22 grammes of the iron or steel to be examined; the smaller quantity is preferable, unless less than .01 per cent. of aluminum is expected. Dissolve the 11 grammes in 44 c.c. strong hydrochloric acid, or the 22 grammes in 88 c.c., on the sand-bath in a 600 c.c. beaker. When dissolved, evaporate to dryness, and redissolve in hydrochloric acid; filter, and wash the silica. Let the total bulk of the filtrate and washings not exceed 200 c.c.

To separate the alumina from the main bulk of the iron, add 3 c.c. of a saturated solution of sodium or ammonium phosphate (of course, free from alumina); then add dilute ammonia till the free acid is neutralized. This point is readily known by a small quantity of phosphate of iron and alumina remaining insoluble after repeatedly shaking the solution. Add hydrochloric acid, drop by drop, till the solution is clear, and then set the beaker on the bath

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to boil. Add 55 c.c. of sodium hyposulphite (of course, carefully tested to be free from alumina), and continue the boiling until the solution does not give off any more sulphurous acid. If there is any doubt whether or not there is an excess of the hyposulphite, add a few more c.c. of acid, and if this gives a precipitate of sulphur sufficient hyposulphite has been added. The smell of sulphur must not be mistaken for sulphurous acid, as the former is always present. One hour's boiling will eliminate all the sulphurous acid. Filter on washed filter-paper (washed with hydrochloric acid), and wash with boiling water to free it from all soluble iron and other salts. Dissolve all the soluble matter out of the precipitate on the filter by pouring over it 5 c.c. of hydrochloric acid and 5 c.c. of boiling water, allowing the solution to collect in the beaker in which the alumina was originally precipitated. Wash out of the filter all soluble matter with as little wash-water as possible. The matter left on the filter should be all sulphur and may be thrown away.

The solution must now be transferred to a platinum dish from the glass beaker, and evaporated to dryness over a beaker of boiling water. When dry add 2 grammes of pure sodium hydrate, free from silica (great care should be taken with reference to this reagent to see that it is perfectly free from alumina), place the sodium on the bottom of the dish, then add about 1 c.c. of boiling water to partially dissolve the hydrate of soda. Place the dish over a flame of a Fletcher's rose-burner until the mass is in a state of tranquil fusion. Allow to cool, and add 50 c.c. of boiling water, and place the dish over a Bunsen burner and boil for 5 minutes. More water must be added, so that the total bulk measures exactly 110 c.c. Filter off the insoluble contents through a dry filter-paper which has been previously washed with acid to remove all alumina. The first portions filtered will be unclear, and must therefore be refiltered.

When 100 c.c. (equivalent to 10 grammes of the 11 of steel taken in the first place), have passed through, measure off that quantity exactly and reject the insoluble matter and the remainder of the solution. Neutralize the caustic soda with hydrochloric acid until the solution reddens blue litmus paper. Add 3 c.c. of sodium phosphate and then hyposulphite of soda in large excess. Boil till all sulphurous acid has been expelled, and add 3 c.c. of ammonium acetate, boiling for a few minutes longer, and filter through a washed filter. Wash well with boiling water until the washings are free from chlorides, and ignite and weigh.

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The precipitate is found to consist of $AlPO^4$, containing 22.36 per cent. of aluminum.

Determinations can readily be made by this method in 12 hours' time. Very accurate results can be obtained by this method, as described in the *Journal* of the Society of Chemical Industry, dated December 31, 1889 (Vol. viii., p. 965).

Especial care must be taken that all reagents are pure, and that the filter-papers are washed free from alumina.

Methods of Preparation of Alumina, the Ore of Aluminum.

Aluminum is now manufactured from the oxide, alumina, which is obtained from the mineral bauxite, this being purified chemically from silica and iron. Bauxite is found in considerable quantities, and fully as pure in quality as the best foreign mineral, in the States of North Carolina and Georgia; and there are vast deposits of it in Ireland and northern France. The average composition of bauxite is about as follows:

Table XXIII.

Analyses of Bauxite.

	Al^2O^3	SiO^2	Fe^2O^3	H^2O	TiO^2	$CaCo^3$
*White from Beaux, France	58.10	21.70	3.00	14.00	3.20	trace
*Brownish-red from Revest, France	57.60	2.80	25.30	10.80	3.10	0.40
*Oolitic from Allsach, France	55.40	4.80	24.80	11.60	3.20	0.20
†White from Georgia	58.50	5.20	2.30	30.10	3.25	0.30
†Reddish-brown from Georgia	58.70	4.80	20.50	13.20	2.10	0.45

The two methods of purification of bauxite are as follows:

Bauxite, or a rich clay, chosen as free from iron as possible, is roasted at a low red-heat, and afterwards is treated with sulphuric acid, which combines with the alumina present, forming sulphate of alumina. This is readily dissolved by water, leaving the great bulk of silica and iron behind. The solution of sulphate of alumina is allowed to settle, the supernatant liquid siphoned off into an evaporating tank and evaporated to dryness. The dry sulphate of alumina is calcined at a red heat, driving off the sulphuric acid, leaving as a residue anhydrous alumina. This calcination seems to be as easy as the calcination of alumina hydrate, and there appears to be

* Authority—Dana.

† Authority—Hunt and Clapp.

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no difficulty in condensing the volatilized sulphuric acid, which can be used over again. This process is easier, on a laboratory-scale, than the soda carbonate method, which is about as follows:

Bauxite is fused with carbonate of soda in a reverberatory furnace. The fused mass is lixiviated with water, which dissolves aluminate of soda, which is decanted off. The solution of aluminate of soda is decomposed by carbonic acid gas, which forms carbonate of soda, which remains in solution, and the alumina hydrate is precipitated. This alumina hydrate is afterwards washed repeatedly with water, dried, and calcined at a red-heat for a considerable time, anhydrous alumina being the result.

UINTAITE, ALBERTITE, GRAHAMITE, AND ASPHALTUM DESCRIBED AND COMPARED, WITH OBSERVATIONS ON BITUMEN AND ITS COMPOUNDS.

BY WM. P. BLAKE, NEW HAVEN, CONN.

(Washington Meeting, February, 1890.)

I HAVE not before had the honor of offering to the Institute a communication on the subject of the variety of asphaltum which I described as uintahite; but several disconnected notices of it have appeared in the *Transactions*, and much additional information regarding it and its relations to other varieties of the asphalt group has been gathered together.

Dr. Henry Wurtz has also discussed the nature of uintaite, and has endeavored to show that it should be considered a variety of grahamite.*

We have also had an interesting paper upon "Asphalt and its Uses," from Captain F. V. Greene,† containing much important technical information and a scheme of classification of the varieties of bitumen and of bituminous compounds.

The object of the present paper is to bring the substance of these disconnected notices together; to present some additional facts; to revise the name uintahite; to compare the substance with the allied bitumens or asphalts, known as albertite and grahamite; and fur-

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* *Engineering and Mining Journal*, August 10, 1889, xlviii., 114.

† *Trans.*, xvii., 355, October, 1888.

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ther to give a general view of the classification and nomenclature of this class of bodies.

Historians record the use of asphalt in the most ancient structures of which we have any knowledge. The Babylonians used it as a cementing substance for their bricks, and according to Brongniart applied it hot. The cradle of the Babylonian Moses was cemented with it. Xenophon, four hundred years before Christ, describes the wall of Media as built of burnt bricks laid in asphalt.* The Scriptures record the use of bitumen in the building of the ark and in the construction of the tower of Babel.

The Latin word *asphaltum* is believed to be derived from the Greek word Ασφαλτος, *asphaltos*, which in turn is considered to be of Phœnician origin. It was used by Homer, and probably in its Latin form gave the name Asphaltites to the lake in Palestine, now known as the Dead Sea, and anciently as the source of bitumen, called *bitumen judaicum*.

Pliny describes this bitumen, which he says at certain seasons of the year is found floating upon Lake Asphaltites, as a "substance which is peculiarly tenacious and adheres to everything it touches."† In the more formal description‡ he considers it as nearly approaching the nature of sulphur, which in some places assumes the form of slime and in others that of an earth. He also describes, under the name of *maltha*, an inflammable mud from the Euphrates.§ This name, which has been generally given to the soft viscid and semi-fluid bitumens, is derived, according to Dana, from the Greek word meaning soft wax, and sometimes a mixture of wax and pitch. Pliny also used the word *pissasphaltus*, derived from Πισσασφαλτος (Dioscorides ||), meaning pitch-asphalt.

Hill, the translator, in the last century, of Theophrastus's "*History of Stones*," thinks that this ancient writer, the pupil of Aristotle, about three hundred years before Christ, referred to asphaltum when he described certain "brittle stones," as he terms them, which became, "as it were, burning coals when put into a fire and emitted, troublesome and disagreeable smell."¶ This description would, however, refer equally well, or better, to coal, and is so applied by Dana.

It will be noted that Pliny describes the bitumen of the Dead Sea as a viscous body, and in another part of his *Natural History*

* *Anabasis*, Book II., iv., § 12.
† *Natural History*, Book vii., § 13.
‡ Book XXXV., § 51.
§ Book II., § 108.
|| *Dana's Mineralogy*, 5th edit., 728.
¶ *History of Stones*, Section XXIII., and notes by Hill.

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he describes how it may be cut asunder by a string if dipped in a peculiar bloody menstruum. In contrast with this idea of a viscid body the following is from Bergman, of Sweden, who, in 1784, gave an interesting description of bitumen. He said:

"The asphalte, or bitumen of Judea, so named because it is found upon the waters of the Lake Asphaltide, in Judea, is a bitumen, solid, black, brittle, brilliant in its fracture, which is vitreous; its black color is really a brownish red, since a thin flake held between the eye and the light appears red. It floats upon water, and in the fire it burns, giving off the odor of amber. When it is very pure it burns with a clear flame without leaving any residue; when impure, it leaves carbon or a scoria."*

This description is more nearly in accord with the characters of pure solid asphalt than any other I have found in the writings of the mineralogists of the last century.

Bergman also describes naphtha, petroleum, maltha and pissasphalte as examples of the gradation of the fluid bitumen into the solid forms of asphalt and of jet, this last being regarded by him as only a more solid bitumen. He also made a distinction between these bituminous substances and bitumen in combination with earthy substances.

Le Sage, in 1777,† had given nearly the same classification of bitumens, placing in sequence, naphtha, petroleum, mineral pitch, maltha and asphalt, or bitumen of Judea, and regarded them all as originating from fluid petroleum. He quotes Wallerius‡ to the same effect.

Wallerius, in 1778, grouped the bitumens under the general head of "Inflammable Substances," placing them in the following order: bitumen, naphtha, petroleum, maltha, asphalte, bituminous earth, stone-coal, jet. Woltersdorff, 1748, under the head of "Bitumens," made two classes—the fluid and the solid—placing petroleum in the first, and amber, succin, mountain pitch and sulphur in the second.

Brongniart, in 1807, under the heading of "Bitumen," credited to Haüy, wrote: "*Les variétés de cette espèce sont la plupart peu distinctes et passent de l' une à l' autre par des nuances insensibles.*"§

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* *Manuel du Minéralogiste ou Sciagraphie du Règne Minéral.* Par M. Torbern Bergman. Traduite et augmentée de notes par M. Mongez. Paris, MDCCLXXXIV.

† Le Sage, *Élémens de Minéralogie*, i., 101–103. 1777.

‡ *Elucet asphaltum nihil aliud esse quam vel petroleum coagulum vel maltham inspissatum.—System. Minéralogie*, ii., 96.

§ *Traité Élémentaire de Minéralogie*, ii., 19–29.

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These varieties are there described in the following order: naphtha, petroleum, maltha, asphalte.

These views of the derivation of this group of hydrocarbons have been followed by most mineralogical writers of this century.

Boussingault, in 1837,* made a special investigation of the composition of asphalt, taking the viscid bitumen of Bechelbronn for his experiments. He found it only slightly acted on by alcohol; the liquid assuming a yellow color when aided by heat, but in sulphuric ether it was easily dissolved. At a temperature of 230° C., in an oil-bath, he separated an oily liquid to which he gave the name *pétrolène*, regarding it as the liquid constituent of bitumens, which, mingled in varying quantities with a solid substance, *asphaltène*, forms the bitumens of different degrees of fluidity. He describes asphaltene as brilliant black in color and luster, with a conchoidal fracture, and heavier than water. Towards a temperature of 300° C., it becomes soft and elastic. It begins to decompose before it melts, and burns like the resins, leaving an abundance of coke.

Boussingault takes the asphalte of Caxitambo, Peru, first brought into notice by Humboldt, as a type of the asphalts, and describes it as having such a large conchoidal fracture and such a brilliant luster and black color that it might be supposed to be obsidian. It has a specific gravity of 1.68. It is difficult to dissolve in petrolene or in the thick oils.

In further support of the opinion of the mixture of the two substances, petrolene and asphaltene, Boussingault says that it is always possible to thicken a soft bitumen by volatilizing the liquid portion by heat. This is done by the Indians of Payta to gain the consistence of bitumen needed for paying their boats.

Dr. T. Sterry Hunt, in 1862, regarded the bitumens and asphalts as intermediate links in the series of substances between solid coal and fluid petroleum. He says: "Anthracite, or nearly pure carbon, on the one hand, and petroleum, or carbon with a maximum of hydrogen on the other, represent the two extremes of the process of which bituminous coals and asphalts are intermediate terms." And in regard to their origin or formation, he says:

"Although we have seen that the solid asphalts, which differ from petroleum in containing less hydrogen and a portion of oxygen, have in most cases been directly formed from organic matters by a process analogous to that which yields coal and petroleum; it appears that the latter body, like other hydrocarbons, may gradually undergo an

* *Ann. de Ch. et de Phys.* (II.), lxiv., p. 141–151.

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oxidizing process, which by removing hydrogen and adding oxygen, at last converts the liquid into substances having the characters of asphalt, of coal, or even of anthracite."

In describing further the occurrence of hard, black, brittle hydrocarbons in fissures of rocks of the Quebec group in Canada, he says: "The mode of occurrence of these matters shows that they have once been in a liquid state, and, as the limestones of this group are in many parts distinctly bituminous, there can be little doubt that the liquid carbonaceous matter was bitumen which has since been slowly oxidized, indurated and converted into these insoluble, infusible coaly and anthracitic bodies."*

Daubrée, writing in 1868,† said that we might include under the general name of bitumens, carbonaceous bodies of varied nature, from petroleum to viscous bitumen, or maltha; sometimes, also, called pissasphalte; and even include that which is solid at ordinary temperatures, to which the name asphalt is reserved.

Beckwith's description of bitumen (asphalt) is as follows:

From whatever source derived, superior bitumen exhibits the following properties: Its *color* is brilliant black with a reddish tinge. The reddish tinge augments when bitumen is softened and drawn into threads. The *fracture* is conchoidal at a low temperature. Below 50° (F.) it is solid and brittle; from 50° to 70° it is elastic and begins to soften; from 70° to 90° it becomes soft and pasty; from 90° to 100° viscous; and at 110° or 120° it melts. Its specific gravity is 1.025. The odor is empyreumatic and free from the noxious smell of coal-tar.

The more modern and comparatively recent attempts at a scientific classification of the bitumens and hydrocarbons generally are based upon their chemical composition. Thus Prof. Dana in the fifth edition of his *Mineralogy* places the hydrocarbons in five groups: 1. Simple; 2. Oxygenated; 3. Acid oxygenated; 4. Salts of organic acids; 5. Nitrogenous; but is obliged to place asphaltum and mineral coal in an appendix to the series.

I now pass to a consideration of the three substances which, under the names of albertite, grahamite, and uintaite, are generally known, and may be considered as the best examples we now have of pure asphalt on a large scale. In regard to albertite, however,

* *Contributions to the Chemical and Geological History of Bitumens and of Pyroschists*, by T. Sterry Hunt, M.A., F.R.S., *American Journal of Science* [II.], xxxv., 164, March, 1863.

† *Substances Minérales.*

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there was, at the time of its discovery, a great difference of opinion amongst mineralogists and chemists as to its true nature, whether it should be classed amongst the coals or with the bitumens and asphalts.

ALBERTITE.

The name, albertite, is credited to Robb, but it was described by Dr. Charles M. Wetherill, in 1852, as a new variety of asphalt.* He was emphatic in his opinion, "that the substance is not coal, nor any variety of coal, but a true and new variety of asphalt." He describes it as brilliant jet-black in color, of high resinous luster; fracture perfectly conchoidal and brilliant; hardness between 1 and 2; density, 1.097; brown color on extremely thin edges. Held in the flame of a candle in such manner as to prevent as much as possible the action of oxygen, it gave evidence of incipient fusion, and adhered to paper when placed in contact with it. He obtained 44.35 per cent. of coke. It was found to be less soluble than Egyptian asphalt, and to bear a great analogy to the asphalt from Caxitambo, as described by Boussingault, the density of which was 1.08.

Melan-asphalt (albertite), analyzed by Wetherill, gave him, in comparison with Cuban asphalt (the ash being subtracted and the oxygen and nitrogen united):

	Cuban asphalt.	Melan-asphalt.
Carbon,	82.670	86.123
Hydrogen,	9.141	8.971
Oxygen and nitrogen,	8.189	4.906

The late Richard C. Taylor was also emphatic in his opinion that this substance should be regarded as asphaltum, and not as coal. He says:

"Like the *chapapote* of Cuba, the New Brunswick mineral has been commonly spoken of as a species of coal, and has been shipped and sold at the ports of the United States and Nova Scotia under that denomination. . . . . After a sufficient investigation of the mine, and of the geological circumstances under which it occurs, there appear to exist no grounds for denominating the mineral contents by any other name than *asphaltum*."†

Mr. Taylor further says, that it has neither a true roof nor floor. It has no overlying or underlying fire clay; no conform-

* *On a New Variety of Asphalt (Melan-Asphalt).* By Charles M. Wetherill, Ph.D. *Trans. Am. Phil. Soc.*, July 16, 1852. 353.

† *Proceedings Amer. Phil. Soc.*, v. 241 (read January 16, 1852).

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able lamination, as in coal or coal-seams, instead of which the divisional planes are arranged transversely, *i.e.*, at right angles to the sides of the vein. It occupies a line of dislocation. It throws off smaller branches which, in places, reunite with the principal vein.

Prof. Charles H. Hitchcock* also regarded the mineral as asphalt, and for the following reasons:

1. The Albert "coal" occurs in true cutting veins, not in sedimentary beds like ordinary coal.

2. The "coal" was originally in a liquid state; was injected into vertical fissures, and subsequently hardened into a substance resembling jet.

3. The Albert coal must be classed with the asphaltic and bituminous veins found in the Quebec group in Canada.

But Dr. Dawson† held a different view, and describes albertite as a coal, or a variety of jet, while at the end he admits that its composition may be nearly that of asphaltene. He says:

"This substance, unfortunately, became a subject of litigation; and as one point in dispute was whether it should be called coal or asphaltum, scientific gentlemen were summoned from the United States as witnesses, and the most discordant opinions were given, both as to the name of the mineral and its geological age. This was not to be wondered at in the circumstances, for the substance was really a new mineral, intermediate between the most bituminous coals and the asphalts. . . . . Some, in all sincerity, called the mineral coal, others asphalt."

Dr. Dawson thought, that in its general characters, with the exception of the color of the powder, it agrees more nearly with the finer varieties of jet, or pitch coal, than with any other substance. Finally, he says: "If we compare albertite, as it has been named by persons not desirous of committing themselves, with the substances most nearly allied to it, we can scarcely avoid arriving at the following conclusions: In its behavior in the fire, chemical composition and electrical properties, the substance is nearly allied to jet, from which, however, it differs in its extreme brittleness, its greater uniformity of texture and more perfect luster and fracture, and also in its black streak; a character which also separates it from ordinary bituminous coal, and all the varieties of asphaltum. Its nearest analogue in this last particular is Lesmahagow coal. Its luster and fracture remarkably assimilate it to the finer varieties of asphalt;

* C. H. Hitchcock, *American Journal of Science*, II., xxxix, 267, May, 1865.

† Dr. J. W. Dawson, F.G.S., *Acadian Geology*, Edinburgh, 1855, p. 198.

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but its streak, mode of combustion, and chemical composition effectually separate it from them. On the whole, the above considerations, in connection with a number of experiments made by Jackson, Hayes and others, and published in the reports on the mineral, place the substance at the head of the pitch coals, or jets, as the purest variety of that species of bituminous coal. It has, however, some claims to be viewed as a distinct mineral species, intermediate between coals and asphalts, and I suspect that its chemical composition may approach to that of asphaltene, the coaly ingredient of the asphalts."

The following description is compiled chiefly from Dr. Dawson's paper:

Structure.—It is free from mineral charcoal and lines of impure coal or earthy matter. It is, however, divided into prismatic pieces by a great number of smooth divisional planes proceeding from wall to wall. Both at the roof and floor it gives distinct evidence of a former pasty or fluid condition, being injected into the most minute fissures of the containing rocks. It seems to occupy a fissure along the anticlinal bend of the strata.

Fracture conchoidal. Brittle, and disposed to fly into fragments.

Color.—The substance has externally an appearance not dissimilar from the ordinary asphalt of commerce in its purest forms. Its color and its powder, or streak, on porcelain, are black, and it is perfectly opaque. My trials show, that while the powder is very dark-colored, it has a shade of brown.

Hardness.—3 nearly (=1 to 2 according to Dana), of Mohs's scale.

Specific Gravity.—1.08 to 1.11, according to Jackson and Hayes.

Fusibility.—In the flame of a spirit lamp it intumesces and emits jets of gas, but does not melt like asphalt. In a close tube, however, it can be melted with some intumescence.

It emits a bituminous odor, and when rubbed becomes electric.

Its deportment with solvents is not given by Dr. Dawson, but according to Dana there is only a trace soluble in alcohol, 4 per cent. in ether, and 30 per cent. in oil of turpentine.

Composition.—Comparative trials were made of its composition and that of a specimen of jet from Whitby, with the following result:

	Albertite.	Jet.
Water,	0.4	1.5
Volatile combustible matter,	57.2	57.1
Coke,	42.4	41.4
	100.0	100.0
Ash in coke,	0 27	4.0

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The analyses by Wetherill have already been given (p. 568) and compare closely with the analyses of uintaite and some of the coals.

GRAHAMITE.

Grahamite, another allied mineral, was first described and named by Dr. Henry Wurtz, in 1869,* from observations made in 1865. This substance was found in a vein some four feet wide in Ritchie county, West Virginia, and it is now exhausted.

I condense and rearrange the chief physical characteristics of grahamite from Dr. Wurtz's descriptions.

Structure.—The structure as shown in mass in the vein is marked by four distinct divisional planes having general parallelism with the walls. The portions next to the walls are coarsely granular with irregular cuboidal jointed cleavage. The next adjoining layers, from 15 to 16 inches in thickness, were highly columnar, with the columns at right angles to the walls. The center of the vein, averaging about 18 inches in thickness, is more compact and massive, and less lustrous in fracture than the other portions. It is described as "clearly *resinoid* in fracture and luster." Fracture subconchoidal, the outer portions of the dike or vein being brittle, fragile, glossy in fracture, breaking partly into small cuboidal fragments, and partly into pencillate fragments, described by Lesley as "pencil-cleavage."

Color.—Streak and powder, dark chocolate-brown.

Hardness.—Not stated.

Specific Gravity.—1.145.

Fusibility.—When heated above 400 degrees Fahrenheit, it softens and undergoes pasty fusion, becomes plastic and may be moulded and can be drawn out into threads. "It endures a temperature far above that of the fusing point of asphalts." Decrepitates and behaves very much like a caking coal.

Solubility.—Indifferent to all acids except strong sulphuric (oil of vitriol) which is colored brown by it. Alkalies do not act on it. Alcohol does not dissolve it. Partially soluble in gasoline and ether; coal-tar benzole and oil of turpentine gradually dissolve most of it. Chloroform and bisulphide of carbon dissolve it rapidly.

Gesner† and others claimed that this mineral was identical with albertite. Professor Lesley considered it to be a mineral pitch or inspissated petroleum, but Dr. Wurtz combated this view and considered the mineral as neither coal, asphaltum nor albertite.

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* *Proceedings American Association for Advancement of Science*, 1869, vol. xviii, 124.

† *Gesner on Coal-Oils, Petroleum*, etc., p. 27.

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In the description of grahamite by Dr. Wurtz, he refers, at the end, to the discovery of a dike-like formation some years before in Colorado, consisting of a substance resembling albertite, and which he determined as grahamite. This, possibly, came from the Uintah region, and may have been similar to, or identical with, the substance I have called uintaite.

Dr. Wurtz also directs attention to the fact, of which I was not aware when my paper was written, that Prof. S. F. Peckham, as Special Agent of the U. S. Census of 1880, reported the occurrence in Utah of veins of a substance resembling the grahamite found in West Virginia.*

Professor J. P. Lesley describes a vein of asphaltum in Wood county, Virginia, which makes a straight course for two-thirds of a mile. Outcrop, 4 feet 10 inches thick, and also 2 feet 6 inches. A shaft had been sunk 34 feet on the vein, which continued of the same thickness, "the asphaltum being filled in, pure and clean," in what "was once, no doubt, an open fissure."†

Dr. J. P. Kimball‡ has referred to grahamite, the asphaltic mineral of the Cristo mine, near Tampico, Mexico. It occurs filling a fissure, and has a "distinct cleavage, a jointed structure as well as columnar partings in the deposits," thus closely resembling the structure of the grahamite of Ritchie, West Virginia. Gravity, 1.156 to 1.167 and 1.176; volatile matter, 57 to 62.14 per cent.; ash, 3 to 5.8 per cent. It was regarded both by Dr. Kimball and by Mr. Wm. Wallace, of Glasgow, not as a coal, but as belonging to the bitumens.

UINTAITE.

The principal facts regarding the brilliant solid asphalt from Uintah county, Utah, to which I gave the name *uintahite*, were first published in the *Engineering and Mining Journal*,§ and later were quoted in our *Transactions*, in connection with a paper by Mr. Jos. M. Locke, on "Gilsonite or Uintahite,"‖ in which he gave the results of chemical analyses, and a still later paper,¶ by Dr. R. W. Raymond, on a "Specimen of Gilsonite from Uintah County."

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* Census Reports, vol. x., p. 20. † *Proceedings Amer. Phil. Soc.*, ix., 185, 1863.

‡ On a Deposit of Grahamite known as the Cristo Coal Mine, etc. New York, May, 1876.

§ Uintahite. A New Variety of Asphaltum from the Uintah Mountains, Utah.—*Engineering and Mining Journal*, December 26, 1885, xl., 431.

‖ *Trans.*, xvi., 162. ¶ *Trans.*, xvii., 113.

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Since these descriptions and notices of uintaite, it has ceased to be a mineralogical curiosity and has become the basis of a considerable trade and industry. The anticipations of its utility in the arts have been more than realized. The deposit has been opened and worked, and companies have been organized to introduce and utilize the article, which is recognized as a superior form of asphaltum.

The first company was formed in Salt Lake City, and the shipments by it for the year 1889, up to the month of September, are reported as aggregating 3,000,000 pounds = 1500 tons. Since then the new company, organized in St. Louis, has shipped 1,200,000 pounds, making a total of 4,200,000 pounds for the year 1889.

The mines are in Uintah Valley, near Fort Duchesne, about 100 miles from Price, a station on the Rio Grande and Western railway. The vein was located as the "Carbon Lode," and was found to be upon one corner of the Uintah Indian Reservation, and special legislation was necessary to have it segregated. The vein is described as a regular fissure, cutting across the country for 12,000 feet, or more, in length, and from 3 to 4 feet thick. It has been opened by fourteen shafts or pits, aggregating 700 feet in depth. The surface-portions of the mineral are fissured and cracked by weathering, are more or less contaminated with sand and earthy matters, and are regarded as of second quality, while the portions below are compact and pure.

In regard to the occurrence we have also the statement of Mr. Wigglesworth, Chief Engineer of the Colorado and Utah Midland Railway,* that it is found about three miles east of Fort Duchesne, near the line between the Uintah and Uncompahgre Indian reservations, in a vein from three to five feet thick, nearly vertical, cutting through nearly horizontal beds of sandstone. Its course can be followed by the eye for a considerable distance on the ridges. Prof. E. W. Claypole writes that it has been followed for about two miles and traced downwards for about 100 feet, still retaining its thickness.†.

I am also informed by a correspondent of the discovery of another vein, about 33 inches thick, which has been traced for the length of three locations, or 4500 feet.

Other localities are cited by Mr. Wigglesworth in the paper to which reference has been made.

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* Note on a Specimen of Gilsonite from Uintah County, Utah, by R. W. Raymond, *Trans.*, xvii., 113.

† *American Geologist*, December, 1889, iv., 6, 386.

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Structure.—The pure mineral, so obtained, breaks with a large conchoidal fracture, of which it presents perhaps the very best typical examples. As Boussingault observed respecting the chapapote asphalt, it might at a distance be mistaken for blocks of obsidian. There is no trace of structure. It appears to break with equal facility in any direction, in this respect differing from albertite in its columnar condition, and from the so-called pencillate structure of portions of the grahamite vein. It should, however, be remembered that we have not had as yet any description, in detail, of the appearance of uintaite in the vein. It is very brittle and is easily crushed to powder.

Color.—Black, brilliant and lustrous; streak and powder a rich dark-brown with a shade of red. It is lighter in color than the streak of albertite, which approaches black.

Hardness, 2 to 2½, not appreciably different from albertite.

Specific Gravity.—1.065 to 1.070.

Fusibility.—Disregarding any possible change of composition, when heated, uintaite may be said to fuse readily in the flame of a candle. Its deportment is like that of sealing-wax or shellac. It can be used in the same way, and takes a sharp impression from a seal. It is plastic while warm, but is not sticky and adhesive. It can be repeatedly melted and cooled without any apparent change of composition, though doubtless losing its more volatile portions when heated.

In my original description of uintaite I pointed out that it differed from albertite and from grahamite in its easy fusibility. No direct comparison of the substances was then made, but the descriptions cited in Dana's *Mineralogy* were taken. It is there stated that albertite differs from ordinary asphaltum by its very imperfect fusion when heated, and that in the flame of a candle it shows incipient fusion. Grahamite is stated by Dr. Wurtz to melt only imperfectly and with a decomposition of the surface, but in this state the interior may be drawn into long threads.

Since then comparison has been made directly with a specimen of albertite obtained many years ago, but which was not accessible when the uintaite was described. This albertite, to my surprise, appears to be nearly as fusible as uintaite, and the two substances do not so greatly differ in this respect as, from the descriptions of albertite, I was led to believe. Fragments of each placed in a glass tube and warmed over a candle fuse quietly, flow in a smooth rounded mass, and by turning the tube each substance coats the

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sides and fills the tube, and on cooling, each hardens to its original condition. There is no appearance of decomposition, and no distillate of oil. The odor becomes stronger, but there is no sensible change of substance.

Albertite and uintaite may thus be said to be equally fusible. Both become rounded smoothly on the edges when brought near the flame of a candle, but their deportment is somewhat different. Albertite at once gives off a slight vapor which is very inflammable and takes fire before contact of the fused surface with the flame, while uintaite does not take fire until brought into the edge of the flame. The odors given off by the two substances are different.

There would seem to be some misapprehension of what constitutes fusibility in these substances, for in addition to the opposite statements already given, Mr. Locke, in his paper upon uintaite, says: "Heat softens gilsonite without melting it."* I can only surmise in this instance he referred to some artificial mixture or possibly an impure uintaite containing earthy substances.

Solubility.—Uintaite readily dissolves in melted wax, in ozocerite, in spermaceti, stearine, etc., and in crude petroleum. Spirit of turpentine, aided by heat, readily dissolves it. Under the trade name of gilsonite, it is now largely used for the manufacture of varnishes. Thus Locke has pointed out that this substance, mixed with turpentine, in the proportion of 1 pound of gilsonite to 5 pints of turpentine, and heated, gives an excellent japanning varnish.†

Uintaite, like albertite and grahamite, is not readily attacked or dissolved by alcohol. The heavy oils, fat, wax and ozocerite unite with it freely.

Composition.—Since the original publication of my description, two or more analyses have been made, and are here brought together. Analysis I., by Messrs. Fristol and Lawver;‡ II., by Prof. T. M. Drown;§ to which I have added for comparison an analysis III., of torbanite, calculated without the ash:

	I.	II.	III. (Torbanite.)
Carbon,	78.43	80.88	80.39
Hydrogen,	10.20	9.76	11.17
Nitrogen,	2.27	3.30	1.32
Oxygen,	8.70	6.05	7.12
Ash,	.40	0.01	

* *Trans.*, xvi., 164.
† *Trans.*, xvi., 165.
‡ *Trans.*, xvi., 163.
§ *Trans.*, xvii, 114.

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A modification of the original name uintahite to uintite has been suggested by the Secretary, Dr. Raymond,* as a more agreeable form, and I am happy to adopt it so far as to drop the h, making the name uintaite, and to use this in preference to uintahite, especially as it is now not uncommon to contract the Indian name Uintah to Uinta. There appears, also, to be a preference in some places for the name gilsonite, after S. H. Gilson, of Salt Lake, who first brought this mineral into commercial notice, and made some insulating compounds from it, to which, I understand, his name was attached rather as a commercial trade name than as a purely mineralogical or scientific one. While desiring to recognize Mr. Gilson's great service to industry in bringing this substance into use, it seemed to me best to attach a name indicative of the source of the material. The name, gilsonite, has also been registered by the St. Louis company as a trade-mark, and this would interfere with its free and unrestricted use for the same substance if found, or mined, or worked by others.

Dr. Henry Wurtz, in August last, sought to show that uintaite should be considered as a variety of grahamite,† comparing it especially with the glossy variety. There is no doubt of the close alliance of these two minerals; yet there are differences, besides the difference of locality, sufficient to make a separate designation desirable. The points of difference between grahamite and uintaite are probably as great as between grahamite and albertite, and if uintaite should be merged in grahamite, both should disappear in albertite, and that, in turn, into a more comprehensive name, asphaltite, giving to the ancient name, asphaltum, or asphaltites, the common termination in *ite*, to conform to mineralogical usage.

In this view, the three minerals, albertite, grahamite and uintaite, from widely separated localities, would be regarded as the best examples, or types, of the species including the hard glance-pitch and the hard bitumen from Lake Asphaltites, so well described by Bergman.

If similarity, or correspondence of the chemical composition of the three substances, is to be taken as evidence of their identity, then they might equally be regarded as varieties of boghead, or Scotch cannel coal, for, according to the analyses cited (computed without the ash), the composition is nearly the same. We see at once that chemical an-

* *Trans.*, xvi., 162, foot-note.

† *Engineering and Mining Journal*, xlviii., 114.

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alysis fails to indicate the peculiar physical characteristics of these substances, and this suggests that the classification of these and other hydrocarbons, in detail, should be based upon their physical characteristics rather than upon their composition, or their derivatives, the nature of which may be varied according to the processes used for their extraction. If these substances are, as supposed, mechanical mixtures, an endless number of species or varieties might be instituted upon the differences of composition.

Bituminous Compounds.

Besides the series of fluid, semi-fluid, and solid bitumens, all of them in a nearly pure condition, we have extensive natural formations of mechanical mixtures of bitumen with limestone, sand, and earthy matters, constituting a class of substances which have important technical uses. Of these, probably the best known is the asphaltic limestone of the Jura mountains, described by Beckwith as "asphaltic rock," and by General Gilmore as "bituminous limestone, called asphaltic rock."

Asphaltic rock, such as is extensively used in Paris for street-paving, is obtained chiefly from Val-de-Travers, in Switzerland, and from Seyssel, in France. It is a bituminous limestone of natural formation, composed of pure (in a commercial sense) carbonate of lime, impregnated with bitumen. Its color is dark chocolate, nearly black; fracture—finely-grained, granular, without cleavage or structure-planes. At a low temperature it is brittle, and breaks like common limestone. At a temperature of 140° or 160° it begins to disintegrate and crumble into powder, and at the temperature of boiling water to 280° the disintegration is complete; it is then a mass of powder. If, in this powdered condition, it is pressed into moulds, or tamped upon a hard surface, such as a roadway, it becomes hard and solid upon cooling, retaining the form given to it, and, in the case of a roadway, presents a monolithic surface, tough and hard, and to a great degree impervious to water and moisture.

The composition varies generally from 7 to 8 per cent. (Beckwith) 10 to 11 per cent. (Greene) of bitumen, and from 93 to 92 per cent. of limestone. Beckwith says that the rock may be uniformly impregnated, but if it contains less than 6 per cent. of bitumen it is too poor to be worked. The rock may also appear rich in bitumen but contain elements of clay, which, not being impregnated like the carbonate of lime, destroy the homogeneity of the formation. "The

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fissures seen in sidewalks are often attributable to this cause, but the presence of clay is easily detected."*

In 1853, I observed in California a very interesting formation of rocks impregnated with bitumen, one of the most remarkable being a compact siliceous stratum, probably infusorial silica, but charged with bitumen. Similar strata were seen near Monterey.†

There are also extensive deposits of viscous bitumen mingled with sand in California, and also in Kentucky and in other localities.

These mechanical aggregations of lime and bitumen and of sand and bitumen are sometimes described, or carelessly referred to, as "bitumen" or "asphalt," but this is entirely in a trade sense, and is inaccurate, and is not justifiable if accurate reference is intended.

Classification of Bitumen and Asphalt.

The subject of classification and nomenclature, and the usage of authorities for centuries, as herein given in some detail, is more important to us when we consider that the existing classification has only recently been the basis of a discussion in the *Transactions*, and that somewhat diverse views and interests are involved in the use of different terms applied to the same compounds.

I refer to the interesting paper by Capt. F. V. Greene, on "Asphalt and its Uses,"‡ and to the discussion of the classification by Mr. W. H. Delano, of London.

The nomenclature in Paris in 1867, from a practical, not mineralogical, standpoint, is stated by Beckwith as "asphalt" or "asphaltic rock" as applied only to a species of limestone impregnated with bitumen, and "mastic of asphalt," an artificial composition formed of the first reduced to powder and mixed with a small quantity of bitumen.

The classification of bituminous substances by Mr. Leon Malo, the consulting engineer of the *Compagnie Générale des Asphalte de France*, and author of a treatise§ upon asphalt and bitumen, is evidently intended for the grouping chiefly of the various natural and artificial compounds of bitumen, rather than for the bitumens or asphalts in their pure state. It is a grouping for technical purposes

* *Report on Asphalt and Bitumen, etc.*, by Arthur Beckwith, C.E., Washington, 1868. *Reports of Paris Exposition*, 1867, vol. iv.

† *Report Geological Reconnaisance, California*, 1855, pp. 178, 182.

‡ *Trans.*, xvii., 357, 373.

§ *Guide Pratique pour la fabrication et l'application de l'asphalte et des bitumes, par Léon Malo, Paris*, 1877, p. 3.

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under the general title of "Bitumen," but a grouping in which the purer forms of bitumen, such as glance-pitch or asphaltum, albertite, grahamite and uintaite have no place or mention. This tabular grouping is as follows:

Tableau Synoptique des Matières Bitumineuses.

Bitume.	1° à l'état libre:	liquide	huile de naphte.
			pètrole de Gabian, de Baku, d'Amerique, etc.
			diverses huiles de schiste.
		visqueux	fontaine de Poix (Auvergne).
			bitume provenant du lavage des sables bitumineux.
			brais de gaz.
	2° mélange à une gangue terreuse:		bitume terreux du Mexique.
			" de Cuba.
			" de l'île de la Trinidad.
	3° mélange à une gangue quartzeuse:		sable bitumineux de Pyrimont-Seyssel.
			" de Clermont.
			" de Bastennes, etc.
	4° imprégnant des schistes:		schistes bitumineux d'Autun.
			" de Buxieres-la-Gaur (Algier).
			" du Dauphiné, etc.
	5° imprégnant des calcaires (asphaltes):		asphalte de Seyssel.
			" du Val-de-Travers.
			" de Lobsann.
			" de Chavaroche.
			" de Clermont, etc.

Reference may be made to the paper of Capt. Greene for a modified form of this classification, in which the solid forms of bitumen are added, thus making a just and needed recognition of asphaltum, and of its separation from the asphaltic limestone. He, however, uses the name bitumen in a generic and comprehensive way to cover and include the whole series of bituminous or asphaltic compounds, both pure and mechanically mixed, giving the name asphalt only to the pure mineral known as asphalt. Mr. Delano prefers to use the name asphalt in a commercial way for the asphaltic limestone, such as is used for paving in Paris. He gives, however,* the following nomenclature as existing in Europe for the past five and twenty years:

1. Asphalt-rock (bituminous limestone).
2. Compressed asphalt for roadways.
3. Asphalt-mastic.
4. Gritted asphalt-mastic.
5. Refined bitumen.

* *Trans.*, xvii., 374.

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This is clearly technical rather than scientific, yet in the first definition there is the recognition that accuracy requires a distinction to be made between asphalt *per se* and asphalt mixed with rock. Careless use of the terms, and the tendency to abbreviate, have doubtless led to the too frequent substitution of the word asphalt for asphalt-rock, rock-asphalt or asphaltic rock, all of which are in use by authorities. Mr. Delano occasionally uses the word asphalt simply when referring to the asphalt-rock. Beckwith sometimes uses the single word asphalt to denote asphaltic rock, and he also describes pure asphalt as bitumen.

We have seen, by the authorities cited in the first portion of this paper, that the use of the word bitumen is generic rather than special, and also that it is synonymous with asphaltum, both of these words being applied to the series of pure or nearly pure substances, ranging from the fluid petroleum to solid asphaltum, but the name asphalt being generally reserved for the solid forms. The various natural and artificial mixtures of bitumen or of asphalt are not grouped with the pure minerals.

The use of the name bitumen for the whole series has the sanction of authority and long usage, but it is not sufficiently comprehensive for our present knowledge of the variety of substances grouped under the more exact designation of hydrocarbons.

The classification proposed by Captain Greene is a step in advance of that given by Malo, but it is objectionable in grouping the free or pure bitumens with the various mechanically-formed rock-mixtures under the one general term, bitumen, which should be applied only to the varieties of pure and unmixed bitumens, while the other substances, into which a variable portion of bitumen enters as a constituent, may be better grouped and called by the name of bituminous compounds.

It is also undesirable to involve the subject of classification with any theory of origin as is done in the definition of bitumen, which is called, "the generic name for the entire class of hydrocarbons resulting from the distillation of coal, or other substances from which coal has been formed."* It is better to group the substances as we find them without regard to their origin; and I would confine the term to natural products, and exclude all artificial products, such as coal-tar and mastic, unless for purely commercial or technical purposes.

Capt. Greene is in error in saying that "native mineral tar, or viscous bitumen, has only been found in such limited quantities,

* *Trans.*, xvii., 357.

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and has been so little used, that it has received no specific name."* As we have seen, the names maltha, brea, and pissasphalt, were applied to the viscous bitumens by the early writers, by Boussingault and others.

From the most remote antiquity to our own time, the native bitumens have been classified and described chiefly by their external and physical characters. They have generally been grouped according to their degrees of fluidity, plasticity, and hardness, and not by their chemical composition beyond the general fact of their being hydrocarbons, the exception being the modern but unsuccessful attempts to secure a purely chemical classification.

The incursions of modern chemistry, which have revolutionized mineralogy as a branch of natural history, have here found an open field; and organic chemistry appears to have taken possession of this department of the science, and promises to absorb it in a cloud of organic derivatives from the protean elements. In the hands of the chemist familiar substances are transformed beyond recognition, and if we attempt to follow the chemists alone, we are soon lost in the mazes of organic chemistry.

Educts vary with the materials and processes used; and, beyond a certain narrow limit, a classification based upon the possible educts cannot be satisfactory. Ultimate analysis fails to give any correct idea of the characteristics and properties of these substances. Polymerism confounds us, and formulas mislead even the few chemists whose lives are devoted to their comprehension.

For practical technical purposes, at least, it is sufficient to follow the old system, and group these substances as they appear to our senses, without the aid of the tortures of the furnace and the retort. Following these views, the following classification of the hydrocarbons is suggested. It is followed by a tabular grouping of the natural and artificial bituminous compounds or mixtures.

* *Trans.*, xvii., 357.

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Tabular Classification of Hydrocarbons.

Hydrocarbons.	Bituminous	Gaseous, ...	Marsh gas. "Natural gas."
		Fluid, ...	Naphtha. Petroleum.
		Viscous (Malthite),	Maltha. Mineral Tar. Brea. Chapapote.
		Elastic, ...	Elaterite. Wurtzilite.
		Solid — Asphaltite,	Albertite. Grahamite. Uintaite.
		Solid — Coal, . .	Bituminous coal. Semi-bituminous coal. Anthracite coal.
	Resinous, ...		Succinite (amber). Copalite. Ambrite, etc.
	Cereous, ...		Ozocerite. Hatchettite, etc.
	Crystalline, ...		Fichtelite. Hartite, etc.

Tabular Classification, or Grouping, of Natural and Artificial Bituminous Compounds.

Bituminous Compounds.	Natural.	Mixed with limestone— "Asphaltic limestone."	Seyssel, Val de Travers, Lobsan, Illinois and other localities.
		Mixed with silica and sand— "Asphaltic sand."	California, Kentucky, Utah and other localities. "Bituminous silica."
		Mixed with earthy matter— "Asphaltic earth."	Trinidad, Cuba, California, Utah.
		Bituminous schists, . . .	Canada, California, Kentucky, Virginia and other localities.
	Artificial.	Fluid, ...	Thick oils from the distillation of Petroleum. "Residuum."
		Viscous, ...	Gas-tar. Pitch.
		Solid, ...	Refined Trinidad asphaltic earth. Mastic of asphaltite. Gritted asphaltic mastic. Paving compounds.

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AVALANCHES.

BY B. E. FERNOW, WASHINGTON, D. C.

(Colorado Meeting, June, 1889.)

MINING interests in the Western mountains are very seriously affected by the danger to property and life from destructive snow-slides and avalanches. This is a danger which the miner has largely brought upon himself by an improvident stripping of the timber from the mountain slopes, and, therefore, he might be expected to take an interest in the reforestation of the areas thus devastated.

A canvass was lately undertaken by the writer to ascertain approximately the extent of the damage suffered from this source, but the difficulty of obtaining reliable data has proved too great to permit, as yet, even an approximation. So much, however, is certain: that the loss of life is considerable, and that much property is destroyed almost every winter. True, neither the number of avalanches, nor their destructiveness, nor the amount of loss compares with what Switzerland experiences. This is due not only to the greater precipitousness of the Alps and to different climatic conditions, but also to the fact that in Switzerland a larger number of settlements is exposed to these dangers.

A very complete and interesting official report on the very numerous and destructive avalanches occurring in Switzerland during the season of 1887–1888 has just been published as a contribution to the inquiry, whether and how these dangers can be averted.

Peculiar weather-conditions, in addition to heavy falls of snow, were the reasons for the frequency of the phenomenon during that winter. This report records 1094 avalanches, by which 850 houses were destroyed or severely damaged; 84 persons and 752 cattle were buried; traffic on railroads and on highways was greatly interrupted, and the damage to personal property alone was valued at over $100,000. One of these snow-masses covered about half an acre, with a cone 250 feet in height. During the same year in Tyrol the number of avalanches was 2647. Of these 1355 run over permanent beds; that is, they recur annually, so that they are

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named, like rivers, as it were; and 765 are expected to recur periodically. The loss that winter was estimated at over $300,000.

Some localities are more liable to snow-slides than others; geological, topographical and climatic conditions, and the greater or less extent of forest-destruction, produce this difference in the frequency of their occurrence.

In Switzerland necessity has long ago stimulated the study of the nature of avalanches and the adoption of protective measures; but only lately has a systematic plan for such measures been adopted. All the protective works that have been built during the last few years under an energetic direction of the forest-administration did their duty well. In the valley of the Leventina torrent, where numerous avalanches fell yearly, even the largest, which was of annual occurrence, was kept out in spite of the large masses of snow; and the success achieved in preventing the formation of these dangers in every case has established beyond doubt the importance of reforestation in connection with other measures of protection.

It is probable that measures are frequently adopted in our mountain districts to protect property against this danger. Only one case, however—that of the Madonna Mine, at Monarch, Colorado; Superintendent, H. D. Mitchell—has come to the writer's notice, where systematic protection is attempted by a system of stakes and old stumps, connected by old wire-cables.

The snow-sheds of railroads crossing mountain passes are built, to some extent, as protection against avalanches. Notably is this the case with the Canadian Pacific railroad, the experiences of which with avalanches are interestingly described in a paper read before the American Society of Civil Engineers, by Mr. Thomas C. Keefer, who says the cost of protection against snow—which, it is true, includes measures to keep the track free from snow in general—has reached within two years the handsome sum of $2,900,000. Besides specially constructed "avalanche-sheds," other methods of protection, to be described further on, are used; but they are all directed against the damaging effects, not against the formation, of avalanches.

With increased permanency of the establishments and settlements to be guarded, greater permanency in the protective works will also be needed, and a closer study of the requirements of such problems will be called for.

The object of this paper is to call attention to one of the prevent-

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able causes of avalanches, namely, indiscriminate forest-destruction, and to state the principles which underlie the formation and action of snow-slides, and which must be kept in view in the construction of protective works.

Conditions of Avalanches.

The formation of avalanches or snow-slides depends (1) on the nature of the snow; (2) on conditions of the weather; (3) on geological, topographical and surface-conditions of the ground.

Snow falling at low temperatures is dry and composed of small loose flakes, without coherence, because the dry crystals do not attach themselves to each other. Such snow does not hang on or ball easily; it lies loose, is easily moved by the wind, like sand, and causes snow-drifts; while the wet snow, falling at higher temperatures (snow may fall at 40° Fahr.), which has begun to thaw while falling and contains much enclosed air, packs tightly and shrinks quickly. On the plain the snow moves only by settling and by drifting. On an incline there is a movement, though it may be but slight, within the mass—a settling which at the same time takes a downward direction, following the law of gravity and depending in its amount on the angle of inclination, configuration, and especially cover of the ground; and also on the nature and amount of the snow. If the snow is of small amount and frozen to the soil, of course it cannot move. Otherwise, even if the conditions are not favorable to motion, and the movement may be hardly noticeable, yet it exists within the mass and becomes apparent by the formation of horizontal rifts on the surface of the snow. This movement, with the pressure due to it, is capable of displacing buildings without much injuring them. The more favorable the conditions for motion, the greater becomes this movement, until at last a slide occurs, moving, in larger or smaller masses, more or less rapidly over the incline.

If a large amount of dry snow falls on a steep deforested slope it behaves like a sand-heap, the particles getting into motion for lack of stability and cohesion in the mass; and, moving forward in the direction of the slope, it sets in motion other masses of snow, and presently the slide "pulls out." The heavier particles gradually accumulate near the ground, while the finer particles form a cloud of snow, bursting into the air like dust and settling down only gradually.

This falling cloud, which obscures the mountain view for some

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time, compresses the air to such an extent that it precedes the avalanche like a tornado, followed and pressed on by the latter in hasty flight and exerting an enormous pressure, often over great distances. Sometimes it is felt two or three miles beyond the actual course of the avalanche.

This so-called "flurry" whirls upward to the height of 100 feet or more above the descending snow, preceding the slide and giving to this class of slides their destructive character. The destruction which is due to the snow-masses of such a slide is insignificant in comparison to that of the compressed-air current. The snow is light, and by the fall dispersed and scattered, and may even pass through a thinly-stocked forest without doing much damage. But the "flurry" will snap off and uproot big trees and carry them with rocks, stones and ice over the mountain slope; and the whirlwind produced in front of it has been known to be of such terrific force as to pick up a man and break and dislocate every bone before he touched the ground again, a limp mass, without a bruise or break in skin or clothing. One such air-current preceding an avalanche is reported to have carried a full-grown larch tree bodily over the tower of a prison, lodging it 300 yards beyond and to have laid low timber 200 to 300 feet beyond the avalanche.

The momentum which the loose mass of snow acquires, if not checked, is sufficient to carry it over and across a valley and up the opposite slope 200 to 300 feet.

By the impact at the bottom, where the snow accumulates, there occurs, to be sure, a compacting of the mass; but this, on account of the dry and loose condition of the snow particles, is not very great.

These slides, which seem to be the most common in the Rocky mountains, occur during heavy falls of snow in cold weather, mostly in the early part of the winter. They may be called *dust-* or *powder*-avalanches, or perhaps avalanches proper, in contradistinction to the snow-slides, described further on, and are due simply to the inability of the snow to keep stable in such masses.* In rare

* The dictionaries and encyclopædias do not seem to know the word "snow-slide," which is the term used in the Rocky Mountain region. The word "snow-slip" is used to denote "a large mass of snow which slips down the side of a mountain and sometimes buries houses" (Webster), while "avalanche" is defined as "a large mass of snow, earth and ice, sliding or rolling down a mountain" (Webster), or "falling down a precipice" (Ogilvie). (To avale—to fall, descend, be lowered: old French *aval*—towards the valley.) There exists, therefore, no definite, distinct idea that might belong to the one word or the other exclusively; and the words have been used as synonyms.

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instances they are occasioned by winds after the storm is over, if, perchance, the snow has been able to preserve its loose and dry condition and to keep in position long enough.

If the snow falls when the temperature is not low, it is wet, heavy and compact, and hangs to the soil closely. If not very deep, it remains undisturbed, provided the soil be not wet or slippery and the ground not very steep. The larger the mass of snow and the warmer the temperature, the greater is the danger of a movement in the mass itself and a consequent slide. In such a slide the snow remains compact and does not disperse, except when it falls over precipices. It does not, therefore, exert much pressure upon the air. It flows more like a snow-stream, now sliding, now rolling over and balling together. In spite of its great weight, its velocity is, by reason of its friction against all obstacles on its road, much smaller than that of a dust-slide, and its effects reach over a smaller space. These may be called *ground-slides* or true slides; for the motion is really a sliding one only.

Such slides of wet snow are apt to occur more often in late winter or spring, especially if the snow has fallen on top of a frozen crust of old snow, or when thawing weather occurs with or soon after the snow-fall. The packing of the snow is such that any persons buried in it are rarely rescued alive; it becomes as hard as ice and will yield only to the pick or powder. In fact, specially designed ice-chisels are used on the Canadian Pacific railroad in removing such snow masses.

Lastly, a third kind of avalanche, with which we have nothing to do in this country, except, perhaps, in Alaska and in Canada, is the *glacier*-avalanche, which is formed when large masses of the glacier disconnect themselves from the main flow, and, falling over a precipice, break up into something like a dust-avalanche.

Apart from the nature of the snow, the following conditions are of moment in the formation of avalanches:

Compact rock-formations are less favorable to the formation of avalanches—especially ground-slides—than stratified rocks. The latter offer more opportunity on the side of the dip than on the side of the outcropping (head). Granite and granitic gneiss mountains are therefore less liable to avalanches than slates (including the calcareous clay-slates known in Switzerland and Italy as *flysch*), limestone, etc. Often, however, the angle of inclination on the dip is so much less than on the head as to reverse the conditions above

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stated; but the slides on the head-side usually do not fall into places where much damage can be anticipated.

Rocks easily disintegrated, like the slates, *flysch*, etc., are more favorable to the formation of slides than solid rock.

Very dangerous are steep, stratified rock-faces, from which spring- and seepage-water oozes out, which keeps the surface moist and slippery, or else, when frozen, gives no chance for the snow to lie.

On a rocky and very steep mountain-side the snow, after acquiring a certain depth, must slide for lack of internal stability, especially when dry. During a continued snow-fall several slides may fall from the same place, but they are mostly of small dimensions and little effect.

If the rock has a soil over it, without vegetation, the formation of slides is dependent, in the first place, on the degree of steepness. Yet other factors, like the depth of the snow, the height of the mountain-wall, etc., are of such influence that the angle of elevation may not be made a mathematical expression of the danger.

If the foot of the snow-wall is washed by a brook, or if a spring or other circumstances disturb the continuity of the snow-masses, the formation of avalanches is favored; it is therefore dangerous in places liable to avalanches to open a track, or even to wade through the snow.

A declivity which offers varying angles, or is broken by occasional steps or terraces, offers so many points of support to the snow-masses that avalanches are less liable to occur; roads, ditches or other artificial barriers to an even descent, offer also such points of support.

Most important is the soil cover. The more and the larger the loose rocks, the more in the line of the horizontal they are placed, the more hold and support has the snow.

Vegetation has a varying effect upon the formation of slides, according to the kind of plants that occupy the ground, and their size. A grass cover or turf is favorable to the sliding of the snow. It has been observed in the mountain meadows, where the hay is made every second year, that slides are less frequent the winter after the grass is cut.

Low shrubs and tree-forms offer a better support to the snow, unless their stems are, as in the case of the mountain alder, so elastic that after being pressed down they exert a pressure against the snow, which tends to interrupt the coherence of the mass, when,

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with the aid of the wind or additional snow-fall, the snow may be set in motion.

High timber affords the best protection against snow-slides, and if the mountaineer had not in his ignorance removed and destroyed this protection, many dangers of a mountain home would be avoided. The importance of the forests in this respect was recognized in the Alps centuries ago, and wherever "ban"-forests were maintained immunity from avalanches to the extent of the forest has been secured. An interesting account exists of the ban-forest of Urseren, which was reserved by the community as early as the year 1397, and of the constant fight which was necessary against reckless advisers to keep it intact; until now the beneficial effect is fully recognized and reforestations have been begun everywhere, under preliminary protecting walls or other safety-works, wherever the danger of avalanches exists.

Yet even timber-forest is not always absolute protection against avalanches, since the place of incipient formation of the slide may lie above timber-line. Such avalanches, formed above the forest-line, if small, first break a wedge into the forest below them, but by repeated action the wedge is enlarged, and gradually a road broken through the forest. Large avalanches coming from such heights break, even in their first descent, through the best-rooted full-grown forests of entire mountain-sides, hurling earth, rock and timber into the valley below.

Water, the great mover of the earth, is an active agent in the formation of avalanches. Not only the spring- and seepage-water, but also rain and melting snow-water, exert their influence. Ground-slides occur mostly in warm weather, in the spring, when the snow melts. Then the snow settles and becomes more compact, has a greater specific weight and, therefore, more tendency to slide; the snow-water penetrates through to the soil, and if the soil is not frozen, saturates it, and then seeks to flow off between soil and snow, by which the hold of the snow on the soil is loosened, the latter made slippery, and the sliding facilitated. How soon this influence of water becomes active depends on how soon the soil is saturated with water, and this, again, on the kind of soil and sub-soil.

Clay soils are soon saturated in their upper strata, and the snow-water sooner begins to flow over the surface; a permeable soil, on the contrary, with permeable sub-soil, does not come to the point of saturation at all, and the danger is avoided. If the soil is frozen

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the water cannot penetrate it, and the sliding takes place the sooner. Rain, of course, adds to the water which loosens the snow.

Configuration, as has been said, may prevent the formation of avalanches by presenting a number of points of support. Yet where there are in the ravines sink-holes or troughs in which the snow accumulates, the melting snow-water collects below the snow and loosens the masses, which may thunder into the valley, following the course of the ravine.

Lastly, even any exterior disturbance of their stability may loosen the masses. A stone or an icicle falling, or snow dropping from the branches of a tree, when accompanied by a strong wind, is liable to start the slide. In like manner one avalanche, by the concussion of the air which it produces, may start others in its neighborhood. It has been frequently observed, also, that a sound, as of a gun, of church-bells, of an explosion from mines, etc., may start the snow. This has given rise to the proposal to start the slides by gun-shots before they are likely to become dangerous.

On the other hand, sometimes in the stillest weather the danger may be greatest, as the snow falling during such weather accumulates to a large amount before it breaks loose; while in stormy weather smaller masses are constantly set in motion and come to strand on the terraces, etc., thus reducing the danger. Some special conditions for the formation of slides exist, when days with a temperature above freezing, which thaws the surface of the snow, alternate with cold nights, during which the surface freezes over. If fresh snow falls on such a surface it is most liable to form into an avalanche; and sometimes, if a heavy snowfall occurs, it may break through the crust and carry the lower masses with it. This kind of slide occurs mostly on southern exposures.

As regards season and periodicity of slides the greatest diversity exists. New tracks are opened every year, while in other places the old tracks become scenes of disaster at varying intervals, some yearly, some every half or every full century. It depends, of course, on the amount of fallen snow, in connection with weather-conditions, especially the direction and force of winds. The most regular falls occur in spring during the period of thaws.

In wind-still weather and warm sunshine the time of fall occurs in the first hours of the afternoon; yet, if a warm wind blows at any time of day or night, the avalanches may start.

The ground-slides follow more or less the contour of the ground, like running water. Usually the masses start sliding; later on they

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roll over when passing over steeper and rocky ground, and in ravines they are compressed and compacted.

As in a stream of water, the greatest velocity is found in the center of the slide, where in a straight track the largest mass and the least friction exist. If turned from the straight track the force is greatest on the outer curve, where trees, turf, soil and rocks are torn up and swept away altogether.

Dust-slides occur most frequently during snow-falls at low temperatures, especially in windy weather. They occur oftener in clear weather than when the sky is clouded.

Protective Measures.

Measures of protection against avalanches and snow-slides may be taken with a view of diminishing or averting the damage after the avalanche is started. There are localities where the configuration of the ground is such as to make any other measures futile. But wherever the ground allows it will be found preferable and more serviceable to prevent the formation of avalanches rather than to guard against their effects.

The precautionary measures consist in placing buildings into the mountain-side, so that an avalanche would shoot over the building; providing places of retreat in case of danger; placing stone and dirt-heaps, or protecting walls with an acute angle, against the track of the avalanche some distance above the houses; or, to protect mountain roads, erecting on the mountain-side galleries of stone and timber, over which the snow-masses are expected to slide, landing on the other side of the road.

On the Canadian Pacific railroad, in the Selkirks, where avalanches are expected, the line is thrown well into the mountain-side and a special "avalanche-shed" is constructed, the roof of which is conformed as much as possible to the mountain-slope. Where it cannot be thrown in far enough for this purpose, a broad bench of natural ground is left to take the impact of the avalanche and send it in a tangent over the roof of a comparatively light shed. When the ravines are too deep to be filled up, the line is thrown out as far as possible, the ravine bridged with a clear span, the abutments being protected by a glance-crib and split fence, and a highway is made for the avalanche to pass under the track. The typical avalanche-shed is made with solid rock-filled crib-work upon the mountain-side, and strongly braced frame-work for its outer wall. The space between the crib-work and the mountain-side is filled in,

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so as to conduct the avalanche over the roof of the snow-shed without striking heavily against it. These sheds are built of cedar and Douglas spruce, and cost $40 to $70 per linear foot. There are built on the flanks of these sheds, and outside of the main track of the avalanche, extensions or gallery-sheds, which are without crib-work, but have their roofs extended against the mountain-side upon strong frame-work; these are terminated, when necessary, with strong parapets, as much as 10 feet high, to prevent the overflow of the lighter snow from the wings of the avalanche. By means of these parapets 40 to 50 feet of shedding at each end are saved. The same principle is adopted where slides come down narrow ravines, in which case the profile of the roof is a trough the width of the ravine. These are called "scoop-sheds."

There are not less than 25,000,000 feet B. M. of sawed material, and 1,140,000 linear feet of round timber used in this system of protective sheds by the Canadian Pacific railroad.*

In order to prevent the starting of avalanches, and thus effectually and permanently remove the danger, their point of departure must be carefully ascertained by observation. Usually the actual place of departure lies higher than the observed place; because before the rush begins there has been a slow movement of the snow above, exerting a pressure and eventually setting the masses below in motion.

If the place of departure is found on a large, steep, rocky face, there is no use in attempting to secure it; but if the starting-point is situated in a ravine, or on a slope with projecting points and rocks or rock-walls, securing is possible, and the question is simply whether the cost of doing so will be in proportion to the benefit to be derived.

The object of the retaining-works is to interrupt the continuity of the snow-field, and by taking up some of the pressure from the upper snow-masses and supporting them, to prevent the first movement.

The work consists of walls or wood-work, or a combination of both. Ditches are not without influence, but are by no means as effective as wall and wood-work, and present, besides, several objections; they are collectors of water, which is liable to carry away the earth-work; and they are soon filled up or trodden down. Similar objections apply to terraces.

The choice of stone or wood for safety-works depends mainly on the costliness of either. Above the timber-line, stone would of

* For full description see *Trans. Am. Soc. Civ. Eng.*, August, 1888.

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course be preferable; below the line, it is employed only where reforestation is not possible and therefore permanent constructions are required.

Wood structures consist of rows of pile-work, for which, of course, the most durable wood obtainable in the region is used. Split wood is better than round, since it holds the snow better. The piles are usually made 4½ to 5 feet long and have a diameter of 5 to 6 inches. Preserving processes, charring, etc., are unnecessary in the altitudes where this work is to stand. The posts are pointed at one end and driven (with a wooden wedge, so as not to split them), if possible, 2½ feet vertically into the ground, and are secured with stones. They should not be more than 3 feet above ground.

An interval of two feet between the piles is sufficient. It is not necessary to connect them by wicker-work, as the snow does not slide through the piles; but where the ground is thin, or very dry, or very steep, or otherwise lacking in hold, such wicker-work made of branches, may be used to connect and thus to strengthen the piles. The two end-piles must be especially secured. The length of the rows of piles and their distance from each other depend upon the configuration and the angles of inclination of the ground.

The choice of the place where the row of piles is to be put is of importance. The work is begun at the top or starting-place of the avalanche and progresses downwards. The piles must be set in the horizontal line, so that the pressure of the snow will be evenly distributed over the whole line. They are also to be placed where a change of fall (angle of inclination) from a less steep to a steeper incline occurs. If no natural terraces are found, such may be made by cutting into the slope above the piles, making a terrace of two feet in the side of the mountain. The ground thus removed is placed on the lower side of the piles and covered with turf.

Where piles cannot be driven, "snow-bridges" may be constructed. Where narrow rills or runs are to be protected, a tree is thrown across and secured at its end against other trees, or by posts, or placed upon trestles, if necessary, supported in the middle by posts. Over this tree are placed sticks, with a slight inclination to the mountain side, somewhat in the shape of a corduroy bridge, and fastened with wooden nails to the tree and secured by stones on the ground, if possible.

The retaining works of stone are the strongest and most lasting, and are possible everywhere, but are also the most expensive. For a foundation a space of about 3 feet is levelled, with a slight inclina-

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tion into the mountain-side, as if cutting out a road; the base must be natural, not made, soil. The proper making of this foundation-base is of the highest importance. The height of the wall must be 3 feet above the ground on the upper side; the steeper the ground, therefore, the higher necessarily becomes the wall-work. Simple dry walls are all that is needed, which in addition to being cheaper, permit rain and snow-water to seep through.

Especially the ends of the walls must be well secured, and the use of cement in this part of the wall is recommended. The roof is made of large slabs, or, in their absence, of sods.

The length of walls, as of pile-rows, depends on the configuration of the soil. The distance of the walls from each other may be greater than with wood-work.

Iron rods, let into the rock and covered with wood-work, have also been used. The choice of work and its adaptation to the configuration of the ground is of the greatest importance.

If the slope presents an even surface, like a roof, the pile-rows and walls are placed at intervals, so that each next following series covers the open spaces of the one before.

In ravines, the work is to be placed where the slope changes to the steeper wall. Large bowlders or rocks rising above the surface are often the incipient cause of slides. The safety-work is to be placed at their foot, so, however, that the snow sliding from such elevations would not fall upon the wall, but in front of it, to be there retained.

Springs and collections of seepage-water must be taken care of and properly conducted away.

A yearly inspection and repairs are, of course, necessary.

To make the constant maintenance of these works unnecessary and to permanently obliberate the danger, reforestation, wherever practicable, offers the only means. This it is true, may be considered under the present conditions of our civilization, too expensive a measure, although the time when forest-planting and forest-management will have to be resorted to and will prove remunerative in the United States, is not far distant.

It is, however, possible to prevent the extension of the danger from avalanches by a more rational use of the natural forest-growth. This consists in avoiding the extensive clearing of mountain-slopes, either by culling gradually and encouraging the natural reforestation by keeping fires and cattle out, or by cutting in horizontal strips of moderate width along the mountain-side, leaving strips of

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timber to hold the snow and to re-seed the cleared spaces. When these have grown up to young timber, the remaining strips of old timber may be removed, leaving only a few seed-trees on the ground.

The damage done by avalanches does not confine itself to the immediate destruction of life and property within their course, but, by tearing up tracts for land-slides and dangerous torrents, which bring down débris and detritus, filling up the lower river-beds, they give rise eventually to destructive floods and endanger districts at considerable distances from the cause of the original disaster. Hence, the question of their prevention is one that concerns the community at large, and brings us to the inevitable logical conclusion, that mountain forests should always remain in the hands of the Government under a rational and conservative forest-administration.

Rescue of Buried People.

It may be of value to those who have to battle with these mountain dangers to add a few hints in regard to the rescue and resuscitation of those buried in avalanches.

In the first place, it is well to know that persons may live as long as 100 hours; an authenticated account of such a case, at least, is found in the official report above cited. The work of rescue should therefore never be speedily abandoned.

Death usually occurs either through bodily injuries or through suffocation, mostly the latter, and this occurs the sooner the more compact the snow; therefore, there is less chance of rescue from the ground slides of wet snow—which packs like ice—than from the dry snow avalanches. The position in which the entombed comes to rest has much to do with his chance of survival. In an erect position, with less snow pressing upon head and breast, there is less danger from suffocation than when the body is in a prostrate position and more deeply buried. When, by a lucky accident, the face comes to lie near one of those hollow spaces which, by the compressed air enclosed, are often formed in avalanches, breathing is facilitated, and the buried person can sometimes save himself; for the heat of the body thaws the snow gradually, around breast and abdomen first, and more slowly around arms and legs, because these parts are more distant from the source of heat.

From these observations some rules for the conduct of those who cannot escape the slide may be inferred. They should try to pre-

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serve an erect position in the fall and to keep the arms near the body, so that the arms may sooner be released by thawing and become useful in the rescue.

When the slide comes to rest, the entombed at first feels a pressure; this is soon relieved, after a few seconds, by the partial freezing of the snow masses and a consequent contraction away from the body.

The entombed hear and understand any noise or voice above them, but their own voice cannot be heard above; this, probably, because the sound-waves cannot sufficiently develop in strength in the surrounding snow.

Rescuing parties should first look over the snow, where the slide has come to rest, for signs of the unfortunate victims. If no parts of the body are visible outside, determine, according to the location and the track which the slide took, the place where most probably the entombed lie. Then let the rescuers take position in line, and with the handles of shovels, or with poles, push into the snow every foot or so, progressing forward as they do so. If a human body is touched the elasticity will be noted; also, the poles pushed down around the body will sink deeper, for the entombed usually do not lie on the ground, but are imbedded in the snow.

In all cases the first duty of the rescuers, when they discover the body in the snow, is to provide means for breathing, by opening up a channel to the mouth.

If the rescued is apparently dead, respiration must first be restored. This is done by placing the body on the belly, supporting the head lightly, pressing evenly and slowly with flat hands upon the sides of the breast, rolling the body over on the side and a little further, and back on its belly, repeating the pressure on the sides; this movement should be repeated sixteen to twenty times per minute.

Another mode is to place the body into sitting posture, supporting the head, grasping from behind the two forearms in the middle, and moving them forward and upward until they touch both sides of the head, then move them down and press with them against the sides of the breast, repeating the process sixteen to twenty times a minute. This treatment should be continued for twenty to thirty minutes, if necessary, while rubbing the patient vigorously with flannel or woollen cloth in the direction from feet and hands towards the body.

Besides absence of respiration, cold and hunger may add to the causes of asphyxia. But even in such cases respiration is first of all

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to be restored. The warming should be only gradual, never in a warm room or with heated bottles, but always by rubbing.

When the body becomes warm the danger in nose, ears, hands and feet of a surplus of blood, or of impeded blood circulation, is avoided by cold compresses, by rubbing with snow, and elevating the affected parts.

As soon as respiration is restored, small doses of coffee, tea or brandy should be given.

When it is considered that in the winter of 1887, *in Colorado alone,* two hundred lives were reported lost in avalanches, it becomes plain, on humanitarian grounds, even if there were no considerations of economy and profit, that the prevention of such calamities is a public duty. The obvious remedy in the rational use and administration of the mountain-forests is entirely practicable. The neglect of it is sure to involve the necessity of costlier remedies hereafter.

THE TREATMENT OF FINE GOLD IN THE SANDS OF SNAKE RIVER, IDAHO.

BY T. EGLESTON, PH.D., NEW YORK CITY.

(Ottawa Meeting, October, 1889.)

THE sands of Snake River, Idaho, have long been known to contain gold. They were worked by some of the first prospectors who came to Idaho, and on the banks still stand the ruins of camps abandoned for years. There are almost always prospectors searching for gold during the season of low water, and it is not an infrequent thing to find a miner of the old type actually panning the sand, or the prospect-hole which he has just left. There is more or less mystery about these miners and their methods of finding the spots where they work. They are always subjects of much speculation to the ordinary traveller. The rich discoveries are, for the best of reasons, concealed until the claims can be taken up. It is difficult to find a place where there are no prospects, but the "colors" are so very fine that they do not offer much inducement to enterprise. The California miner usually values a color at from 5 to 10 cents, often the latter; while on Snake River it would take 100 colors, or even more, to make that amount.

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There is not much difficulty in panning out the heavier pieces of this gravel, so that those which remain have not over half the diameter of a pin's head. Below this size the separation is extremely difficult, as the flakes are very thin. The heavier pieces of the basalt, the black sand and fine gold remain persistently together; and, after the iron has been separated by a magnet, there still remains a material which appears grayish to the eye, has about the same gravity as the light particles of gold, and cannot be removed with an electro-magnet run by a very strong battery. After the heavy black sand has been separated by the magnet, the fine particles of gold float, while the gray sand sinks; so that I had the greatest difficulty in concentrating half a liter of material, taken up from under the head of the burlap sluice, into a bulk of 15 by 4 millimeters. A large part of the concentrated gold floats; and when, after much trouble, the surfaces are wetted and the gold is got under the water on to the top of the sand, the first wave from the other side of the pan over the sand floats the gold again.

The minerals contained in the fine sand are quartz, chalcedony, semi-opal, zircon, topaz and magnetite. I have not been able to distinguish any trace of pyrite, and have only occasionally seen small grains of peridote. These minerals are sometimes ground and broken, and sometimes are small detached crystals, which can, however, be seen only with high powers. The gold itself is in flattened forms. Almost every piece is convex and shows surfaces which are bright and more or less mammillated, having the appearance of having been acted upon by some reagent. They are quite similar to the artificial nuggets which I described some years since.*

It is a question of interest how this gold got into the sands. The most probable answer seems, at first sight, to be that it came from the abrasion of the rocks. This leads to the further question, Why is it, then, so very much finer than gold-sands ordinarily are? I do not pretend to be able to answer these questions decisively now, but hope to do so at some future time. The object of this paper is rather to describe the method in use for catching a part of the gold.

Snake River, during the course of nearly 100 miles, which I have studied, runs through a columnar basalt. This rock covers the plains, where the surface of the flow is covered with great nodules from 15 to 40 feet in diameter, and from 10 to 20 feet high, which have a radiated and at the same time a columnar structure. The

* *Trans.*, ix., 633.

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erosion and destruction of this rock has formed the sands of the great Idaho desert. The rock is much fissured and is covered only to a slight depth with sand. It is generally compact, but the surface of the ground is covered in some places with small pieces so full of bubble holes that they appear like scoria. All such pieces have a more or less large coating of silica on them. It is in this rock that the Lost River disappears, to come out again, as is generally believed, in one of the numerous large springs which flow from the north side of the bluff below Shoshone Falls.

The Oregon Short Line railroad crosses Snake River at American Falls station, where the river is 1500 feet wide and descends 50 feet in several falls 10 or 15 feet high between bluffs 30 to 60 feet high. It runs West in this way a little over 100 miles, the bluffs on both sides of the river gradually growing deeper, until at what is known as Twin Falls the river suddenly falls 175 feet, leaving the bluffs 500 feet high. In the course of the next 3 miles it forms Shoshone Falls, where the river first narrows to 750 feet in width and then takes a jump, first of 75 feet and then, within a few yards, of 210 feet, leaving the bluffs at the foot of the Falls 1200 feet high on both sides of the cañon and the river 1000 feet wide. The top of the bluffs is a very hard compact basalt, with occasional small nodules of chalcedony, but only in a single place, just above the Twin Lakes, below Shoshone Falls, did I see in this top rock any large holes, and nowhere any appearance of weathering or decomposition in the rock itself in place. The rock is in many places coated with a white covering of silica, formed by the evaporation of the water percolating through or collected on the surface. These coatings sometimes take fantastic shapes, as in the case of the Devil's Spade, about a mile above Shoshone Falls, where what appears to be a painting in white of a gardener's spade can be seen in a niche just below the top of the bluff. Notwithstanding this deposition of silica, the rock shows to the eye no trace of decomposition. I was told that iron pyrites is found in nodules in the upper part of the bluffs; but I did not see, in the course of more than a week that I spent on the river, a single piece of iron pyrites, however small, though I looked carefully for it. Below the great Shoshone Falls, on the level of the river, the rock is porphyritic, light gray in color, and much more feldspathic and friable than the basalt above, and thoroughly rotten from decomposition. About 100 feet from the foot of the falls it is so worn away that I crawled through apertures in the decomposed rock for 25 or 30 feet, at a dis-

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tance of about 10 feet above the water, the outside of the rock still being intact. This decomposition is much like that which occurs in the sand-rock of the Kentucky River near Jackson, Ky. It is not an uncommon thing in the various side-cañons which lead into the main cañon of the river to find that the bottom-rock has been so much decomposed, by the absorption of its alkalies, or so much washed out, that pieces of the overhanging hard rock, 50 to 75 feet in length and 8 to 10 feet in width, have dropped down, retaining their vertical position, leaving a chasm 2 or 3 feet wide between them and the main rock. Striking examples of this occur in the side cañon near the Twin Falls and below the Great Falls.

This decomposition of the underlying rock has taken place on a very large scale, and is particularly visible just above and for a considerable distance below the main falls, so that the surface for 500 feet from the main cliff is broken up into what appears at first sight to be small terraces, which are difficult of access. From the top of the cliffs this sinking of the rocks gives an appearance, on a very small scale, quite similar to the grand cañon of the Colorado river, as seen about 75 miles from Flagstaff, Arizona, where the same phenomena take place on a different rock, on a scale probably larger than anywhere else in the world. The water of the river is cool, and slightly alkaline to the taste. The soil is full of alkali, which is undoubtedly one of the results of the decomposition of the rock. It is also probably one of the sources of the solution of the gold, and its subsequent deposition in the sands of the high banks as well as the deposition of the silica minerals formed by evaporation.

It is a fact well known to those who chlorinate pyrites concentrates, that the gold they contain is in all degrees of fineness; that, in some cases, it is extremely difficult to attack the gold; and that, in other cases, the loss in roasting is enormous, on account of the very large surface, as compared to the weight, of the gold particles. It is known that the gold is much finer in some pyrites than in others, and I supposed at first that this might account perfectly for the condition of the gold in these sands. The microscopic examination, however, does not justify the conclusion that the gold has come exclusively from the decomposition of pyrites; since in no case have I been able to find any gold that was coated, or the surface of which was not just as bright as if it had been for a long time exposed to chemicals. That small quantities of gold are contained in solution in the waters flowing through these rocks

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seems to be probable. The experiments that I have been able to make in the laboratory tend to confirm this opinion.

Snake River, below Shoshone Falls, runs for 18 miles through basalt, the height of the bluff varying from 1200 feet at the falls to about 500 feet at the end of that distance. It then commences to widen out gradually, the talus of sand increasing little by little in height until at last the outlines of the rock only, come up at the top of the bluff. These finally disappear; and what was a talus becomes sand-hills on the south side, and, on the north, more or less irregular sand-bottoms, with cliffs of basalt varying in height, but generally not more than 30 to 40 feet high, so that in several places roads have been constructed in the side cañons from both sides down on to the bottoms. On the top of the bluff there is no water. The rock is columnar and very much fissured. It dips south, so that there are no springs on the south side of the river, but all along the north side at intervals of three or four miles great springs gush out, which form at once considerable streams, issuing directly from the rock.

The river bottom-lands widen from nothing to about half a mile. They rise above low-water mark as much as 30 to 40 feet. Where it is possible to get water to them, either from the springs on the North side or by flumes taking water high up the river on either side, the sands, after the alkali is washed out of them, make excellent soil, and in both ways some beautiful ranches have been made.

The sand in the river contains some gold, not in large quantity anywhere. It is so very fine that it is difficult to collect it without special apparatus. The gold value of these sands is estimated at about five cents to the cubic yard, which includes top soil and bottom gravel as well. The pay-streak is richer than this, but in the working it is impossible to make any separation.

The pay-streaks, after prospecting, are worked on both sides of the river. There are a number of these placer-workings about 50 miles below Shoshone Falls, which have been operated for a number of years by a method but little known. One of these, known as Hunt's placer-claim, at Salmon Falls, which is 17 miles across the desert from Bliss station on the Oregon Short Line railroad, I had occasion to visit in the summer of 1889.

The method of excavating the sands is simple. It consists in bringing a ditch into the pay-dirt, and making the ditch itself do almost all the work of excavation. The fall of the water breaks the ground down, and sufficient grade is given to carry it to the machines.

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The best grade for the water-ditch has been found to be 4 inches in 12 feet. Ditches with earthen banks, and flumes, sometimes partly in earth, and at others wholly on trestles, are used. When the ground-sluice is not over 36 inches wide, gravel runs easily on a 4-inch grade. The long flumes have a grade of 3½ inches in 12 feet. The ground-sluice is made so as to work gradually back, cutting itself out by the fall of the water. The top soil is broken back by the pit-man with a steel bar, and the lumps are broken up with a light pick. The pay-gravel varies from 10 to 25 feet in thickness, and has to be washed in benches, taking from 5 to 8 feet for a bench. No ditch is required in the gravelly places. The stream is directed around the bank by the pit-man, who uses "shear boards," 14 inches wide and 2 inches thick, with round sticks 1 foot long and 2 inches in diameter, run through them as handles. These are so placed as to direct the water against the bank, to undermine it slowly, so that the sands cave, but not so rapidly as to impede the course of the water. These planks require the constant attention, in the day time, of one man in the pit, who in ten hours will bar down ground enough to keep the machine working for twenty-four hours. The skill of the pit-man consists in so barring his gravel and arranging his shear-boards that the pit will take care of itself at night. The current must be strong enough to carry all that falls into the sluice leading to the machine. Occasionally, the ground-sluice is dammed by the gravel in the night, and cuts a channel in such a way as to make it impossible to reach certain blocks of ground. This is not, however, a serious inconvenience; since the ground has to be broken in benches, and the spot so left is taken in, in working the bench below.

The stream carrying the broken-up gravel runs through sluices, in general not more than 24 inches wide. But, from about 8 feet before it reaches the first iron plate of the machine, the sluice is enlarged, so that where it meets the plate, it is 4 feet wide. The machine consists of a board-sluice, Figs. 1 and 2, lined with perforated sheet-iron plates, 3 by 4 feet in size, which are called "grizzlies." Below them, on both sides, is an inclined sluice, leading to an under-current, which discharges into sluices covered with burlap, called "sack-boxes." The width of the machine is constant; but the length varies with the fineness of the gold to be caught. At Hunt's claim there are three—one 24, one 30, and one 36 feet long. The flume that carries the gravel is 1 foot lower than the top of the "grizzly." The sand accumulates here and makes a pavement, and prevents the wearing of the bottom. In some of the small machines this sluice is

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paved for some distance with wooden blocks placed on end; but all of them have blank plates for the length of one plate, at least, set almost

FIG. 1.—Plan.

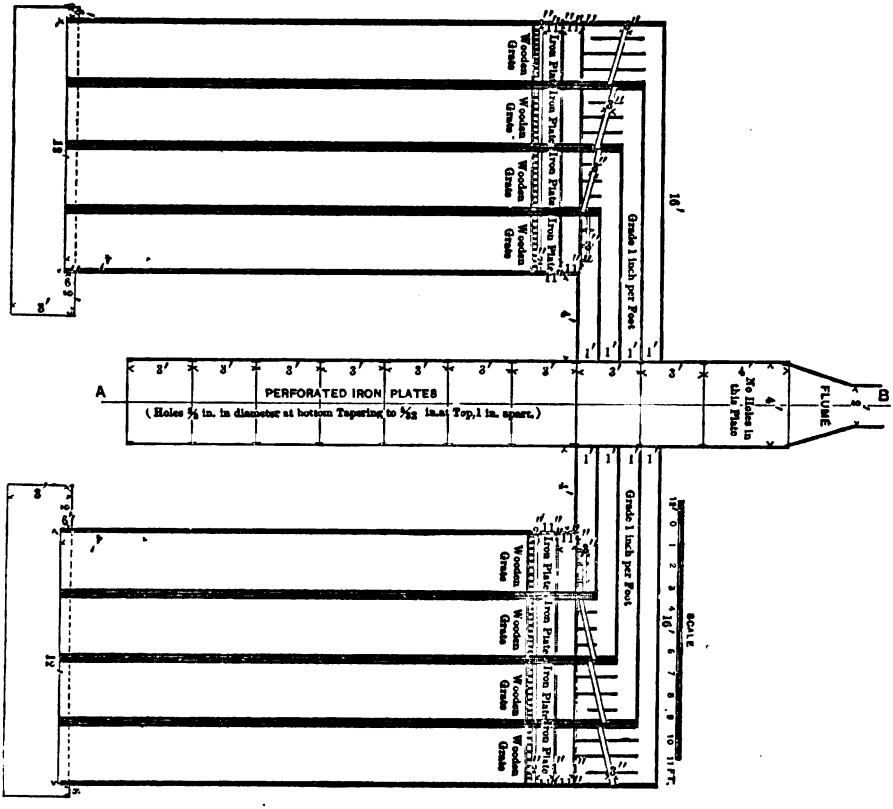
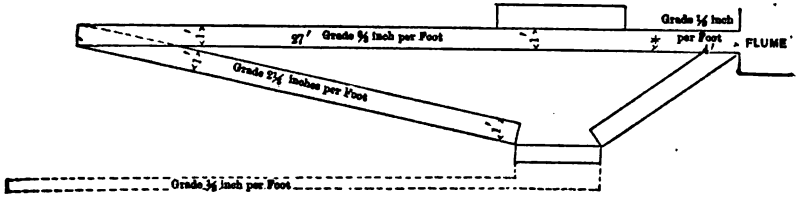


FIG. 2.
Section through A B, FIG. 1.



Plan and Section of Apparatus for Saving Fine Gold.

without grade; and these have been found to wear best, as the water builds its own grade from the sluice to the machine. The blank

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plate is 4 feet long. It is made, like the other plates, of steel or coke iron, $\frac{5}{32}$ of an inch thick. The grizzly-plates are perforated with conical holes, $\frac{1}{2}$ an inch between centers, which are $\frac{5}{32}$ on the top and $\frac{3}{8}$ on the bottom, to prevent the holes from clogging. These plates are 3 feet long and 4 feet wide. The sides of the sluice are made of boards, 1 inch thick and 12 inches wide, except just over the undercurrent, where they are 2 feet wide. The grade of the "grizzlies" varies from $\frac{1}{4}$ to $\frac{3}{4}$ of an inch, according to the size of the gravel. The coarser it is, the more grade is required. In setting up the machine, a frame is first built, and on this the grizzly is placed loose, so that the grade can be changed if necessary by wedges placed underneath it. Ordinarily, $\frac{1}{2}$ an inch to the foot is found to be the best grade. When the grade is determined, the grizzly is wedged firmly to its place. At the end the grizzly discharges the material too large to go through the holes into the tail sluice, which must have sufficient grade to carry off all the tails. Below the grizzly the sides incline both ways towards the center at the rate of 2 to 3 inches, or more, to the foot, and end in an undercurrent called a "sand-tank," which is 4 feet wide. This has a grade of $1\frac{1}{2}$ inches to the foot, towards each side. It is generally placed immediately under the center line in the middle of the machine, but its exact position depends on the lay of the ground. In its center is a sump, 4 inches lower than the lead-boxes, so as to distribute the water equally to the burlap-sluices on each side of the machine. These are 4 feet wide, incline $\frac{1}{2}$ to $\frac{3}{8}$ of an inch to the foot, and are arranged according to the length of the grizzly. For the 36-foot grizzly there are twelve of them, six on a side; for the 30-foot, there are ten, five on a side; for the 24-foot, there usually are only six, three on a side. Four would work better. The grade of the lead-boxes is $\frac{1}{2}$ to 1 inch to the foot. The lead-sluices are arranged, so that the back one carries the gravel to the sluice furthest away from the machine. They are 10 inches wide. In order to distribute the sand evenly in the width of the sluice, there are four divisions at their head, which vary in length so as to distribute the current equally. At their upper end there is a movable grating made of strips of wood, set inclined, to keep out any floating matter which may accidentally get in, and to further distribute the stream. The sluice-boxes are 3 feet wide, and 24, 30 and 36 feet long; made of thin boards placed end to end. The sides are 6 inches high and are nailed to the sides of the bottom boards. They have a grade of $\frac{1}{4}$ to $\frac{3}{4}$ of an inch to the foot. If the material treated is all sand, which is the usual case, the grade is $\frac{1}{4}$;

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if gravel and sand together, ¾. These burlap-sluices connect with the tail-sluice by a sluice at their end, at right angles to the main sluice, which is 3 feet wide and has a grade of ½ an inch to the foot, over which the burlap-sluices project 6 inches, and which projects 2 feet beyond them. These sluices, like the grizzly, are set on a frame, so that the grade can be changed by wedging, and when once determined on, can be set firmly in place. As the tail-sluices are on both sides of the machine, they discharge in front of the grizzly with sufficient force to move all the tails down the main tail-sluice.

At the head of the burlap-sluice, just beyond the wooden gratings, which regulate the flow of the gravel, is a cross-cleat, under which an iron plate, 1 foot wide and as long as the width of the sluice is fitted. This is made of old grizzly iron. It is held down at the side by a cleat, under which it slips.

The sluice is covered with burlap 40 inches wide. It has been found by experience that burlap weighing 7 ounces to the yard is the best. Heavier and lighter have been used, but are not found to answer so well. The lead-sluices have a grade of 1 inch to the foot; when the plate is put on them, there is a slight drop; and here there is always gold to be seen at the clean-up. Each of the lead-sluices is so arranged that it can be cut off separately. This is done by a gate of wood wrapped with burlap to make it quite tight and, in order to further keep out the water, the burlap, which is to go on to the sluice after the clean-up, is thrust into the lead behind the gate, which prevents the passage of water and at the same time wets the burlap, so as to fit it to be spread out on the sluice. These gate-wrappings, as soon as the clean-up is finished, are hung up on each side of the grizzly, so as to be ready for use at the next clean-up. The burlaps on the sluices are held down by cleats of wood, kept in place by wooden eccentrics, screwed to the side, so that when they are turned down, they hold the cleat in place. These cleats are 1½ by 1 inch and are cut from a 1-inch board. After six years' use some of them are so worn that they have to be turned up on their angles to meet the eccentrics. In the 24-inch sluice there are six of these on each side.

To make a clean-up, the man in the pit comes down to the machine with the superintendent, and the water is turned off from each of the sluices, one at a time, commencing always with the outside sluice. The assistant has a piece of flat rubber set in a handle. He knocks down the eccentrics on both sides of the sluice to be cleaned up, with the handle, so as to loosen the cleats. These are then taken

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out, washed in the next sluice and placed crosswise over it, to be at hand when wanted. The superintendent takes out the iron plate at the top and carefully scrapes off the sands and gold, washing the plate in the next sluice. The assistant takes the burlap from the bottom, brings it up to the center of the sluice, and commences to push with his rubber the material collected underneath the burlap on the bottom from the lower end to the middle of the sluice. The superintendent carries the top of the burlap to the middle, folds it to 18 inches and the width of the sluice, then doubles it over towards the side of the sluice, presses it flat with his feet, and places it in one of the boxes. There are two of these boxes for each sluice. They are 18 inches long, 13 inches wide, and 7½ inches deep, and are lined with tin. The woodwork of the boxes is only knocked together, but with constant use they last four years. When the tin commences to rust through, so that they leak, the boxes are used exclusively for the burlaps. The tight boxes are always used for the sand. While the superintendent is arranging the blankets, the assistant goes to the top and pushes the sand down to the middle, to be put into the other box. When the sluice is clean, the fresh-wetted burlap is taken out of the lead-sluice, the upper end being held by the superintendent and the lower end by the assistant, and stretched with both hands over the sluice, leaving the upper end of the burlap to project over the upper end of the sluice. The burlap is shoved under the cross-cleat by the iron plate, which is cut off at the corners to prevent cutting the burlap; this goes under the cleat and is held fast, leaving about 2 inches of the burlap projecting over the top of the plate. The side-cleats are now taken from the adjoining sluice and put in, commencing at the top. On their hands and knees the two men go from the top to the bottom, pressing the cloth in place, spreading the burlap so that it comes up to the sides of the cleats and sometimes over the top, according as it stretches. They then turn down the eccentrics, put in the grating, and turn on the water. This work, in a single sluice, takes five minutes. The next sluice is treated in the same way, and so on until the clean-up is completed. They are careful to spread the burlap tight and smooth; for, if it is wrinkled or bulges, sand collects under it and less gold is caught. If they are properly put down, only two or three pans of sand will be thus collected which, at the head of the sluice, contains considerable visible gold. Most of the gold, however, is caught in the burlaps.

The burlaps are cut so as to cover the whole length of the sluice

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and lap over the lower end; but they wear, and after some use grow short, so that they are generally eighteen inches to two feet shorter than the sluice. The lead-sluices at the head of the burlap-sluice, which distribute the stream, are keyed in and are kept together by diagonal strips. The grating is put in loose, 6 or 8 inches from the slats.

Nothing but burlap has been found to answer for this work. Ducking was used at one time, but it did not catch the gold so well and rotted too fast. The burlaps are used until they go to pieces from rotting. They usually last two months. When worn out, they are used around the gate. When they can no longer be used, they are dried and burned, and the ashes are panned. In order to prevent the closing of the pores by vegetable matter, the burlaps must be spread out in the sun to dry thoroughly after each clean-up, to destroy the plant-life before they can be used again. They last longest in summer. In the winter they are apt to tear from freezing at the sides. The whole work of the three machines is done by five men,—one man in each pit; the helper, who does other work; and the superintendent, who, with the pit-men, makes the clean-up and "rocks" the dirt.

Over the 24-foot machine, 200 to 250 inches of water are constantly run in 24 hours: on the 30-foot, 350; and on the 36-foot, 400 inches. The work done on each machine averages about a cubic yard of gravel for each miner's inch of water used. The essential requisite is to secure a good dump; unless this is done, the tail-sluices will soon be filled up. Usually, on Snake river the spring freshets wash the tail-sluices out; but for two or three years, on account of the low water in the river, this has not been the case. It is also necessary to set the bed-rock sluice in such a way that there will be plenty of water the year round. In some places it has been set too high, so that when the river is low there is no water to wash with, while there is plenty of water in the river. Not only plenty of water, but free water is essential to the economical success of this process.

The boxes from two of the machines which are at a distance, are cleaned in a shed near the house, half a mile from the sluices. In one which is difficult of access, the rocking is done on the spot. The clean-up house is 14 by 12 feet. On the long side against a window is the washing-tank. An ordinary gold-sand rocker is placed at the end next a window, and the retort-furnace opposite to it. The boxes containing the burlaps are piled next the washing-tank. The sand in the boxes is piled next to the rocker. The

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washing-tank is 10 feet long, 3 feet wide and 3 feet deep. It has a shelf 6 inches wide behind, and one 4 inches wide at an angle of forty-five degrees in front. The burlap taken from the box is washed several times up and down in the tank, where water is constantly running in and out. It is then allowed to fall into the tank; the end being held by the hand, is washed by shaking it on to the front ledge, and is then folded backward and forward so as to make the folds 1 foot wide. It is then thrown on to the ledge behind. The gold sand from the burlap which remains on the front ledge is then washed into the tank, the end of the burlap is brought over on to the front ledge, while the rest remains on the opposite ledge, and the space between the two parts of the burlap washed up and down and again folded on the front ledge. This is repeated twice; so that each burlap is washed three times. The ledge is washed every time to throw the gold into the water. The washing is done at some distance from the discharge-pipe, to avoid the danger of the fine gold floating off. The sand collected at the bottom of the vat is allowed to settle until the water is almost clear. The water is drawn off from above at three plug-holes, at different levels, and the sand collected is rocked. The rocker has a plate 60 inches long by 24 inches wide. The end and sides are turned up 1 inch, so that the inside space on the plate is 22 by 59. The hopper, into which the sand is put, is 13 by 15 inches, and 5 inches deep. The bottom of the hopper is made of a grizzly plate and the constant washing has worn grooves laterally towards the holes. The inclination of the plate is $2\frac{3}{4}$ inches in 2 feet, which experience has shown to be the best. The plate is copper and not silvered, silver-plated plates not having been found to answer. The plate is first washed with a swab soaked in potassium cyanide. Then mercury, sprinkled half way down through a fine cloth placed over the end of a bottle, is wiped over the surface with the cyanide swab. The sand is put in at the hopper and the rocker is rocked seventy-eight times a minute. What comes off the plate runs into a tail-sluice, and carries almost no gold, not more than two dollars to the ton. About seven small coal-shovels full are worked in a minute, care being taken never to clog the holes. This is all that is done with the sand from the boxes. The sand from the tank is mixed with cyanide, the proportion being determined by trial, and allowed to stand two hours before rocking. The moment the rocker is stopped, the end of the amalgamated plate is turned up so as to drain to the back, to prevent the loss of mercury and

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amalgam. Each machine is cleaned twice a week. The 24-foot treats from 78 to 80 tons per day, and the others correspondingly more. The gold collected by this method is very fine,—much finer than any that I have seen, except from some of the Arizona placers.

The machine runs itself, with only one man in the pit. It pays to run sand which has from 50 to 85 colors. These machines collect from thirty-five to forty dollars a day each. The work of the clean-up varies with the size of the machine. With the 30-foot machine, all the work of cleaning up the sluices, washing the burlaps, spreading them out to dry, and rocking, is done in six hours. The 36-foot machine takes an hour longer, and the 24-foot an hour less.

This method is one of the least expensive of all the devices for the treatment of fine gold. It requires but little capital and labor, and the returns well repay the men. It requires, however, free water. By changing the grade of the machine and sluices, and multiplying the number of burlaps, very close work can be done.

I desire to express my obligations to Mr. J. S. Hunt, the owner of this claim, and Mr. C. M. Peters, the superintendent, for the assistance they gave me while studying the method on the spot, and since my return to the East.

NOTES ON FUEL-GAS.

BY GEORGE W. GOETZ, MILWAUKEE, WIS.

(Washington Meeting, February, 1890.)

As is well known, the general use of natural gas in Pittsburgh and vicinity for domestic and industrial purposes has shown the great advantages and benefits to be derived by the use of a gaseous fuel.

In order to ascertain and bring together all the facts relating to a large and economical production, purification, distribution, and consumption of fuel-gas made out of coal, the Fuel-Gas and Electric Engineering Company, Ltd., was organized, under the presidency of George Westinghouse, Jr.

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Interviews with leading authorities, in this country and in Europe, and a careful perusal of the enormous literature on this subject, brought out so many conflicting statements as to show the necessity of experiments, on a large and practical scale, to get accurate data to base ideas upon, and to ascertain the practical difficulties and merits of the respective processes proposed. A large number of experiments were made as to the possibilities of water-gas, illuminating-gas, producer-gas and oil-gas as fuels. A bench of retorts, producers, water-gas apparatus, condensers, scrubbers, meters, a laboratory, and a holder, with a capacity of 50,000 cubic feet, being at disposal, it was possible to test the properties of the different gases obtained, and to make mixtures of them in any desired proportion. All these experiments were controlled by carefully-made gas-analyses.

Interesting experiments on the diffusion of gases in a large holder, when two gases of different specific gravity enter the holder alternately or at the same time, were also made, as well as a large number of tests on the combustion of different gases. Mr. Frederick Siemens's statements on luminous flames and on heating by radiation, were fully corroborated by carefully-made tests; and gas-burning devices, based on the radiation of heat from luminous flames, were developed, which will give an efficiency of 84 per cent. of the original heat-units in the gas, 54 per cent. of which results in radiant heat, and is reflected on the floors of a room.

Producer-gas is undoubtedly the cheapest fuel-gas that can be made, and answers all purposes where the air necessary for its combustion can be heated by the off-going heat of the furnace. Tests have shown, however, that producer-gas carries too much nitrogen to permit its distribution to a community, since, in consequence of the presence of this constituent, its calorific power is too low, and its flame extinguishes so easily as to be dangerous for household use. It was important to know how many cubic feet of producer-gas 1 ton of coal will give. Conflicting statements as to this point will be found in the literature of the subject, the figures ranging from 150,000 to 250,000 cubic feet per ton. To get at the actual figures by a practical test, a ten days' run, night and day, was made. Pittsburgh coal of the following average composition was used:

	Per cent.
Water,	1.26
Volatile Matter,	36.22
Fixed Carbon,	57.98
Sulphur,	0.70
Ash,	3.78

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Fig. 2.

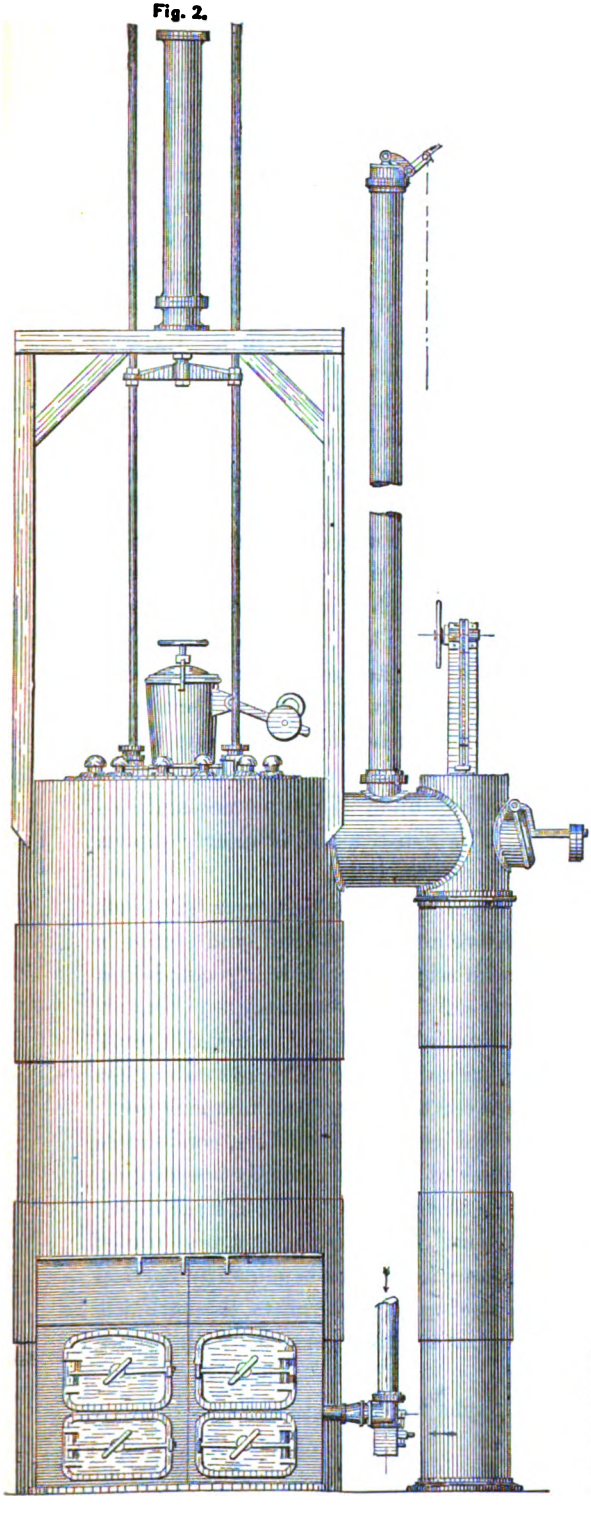
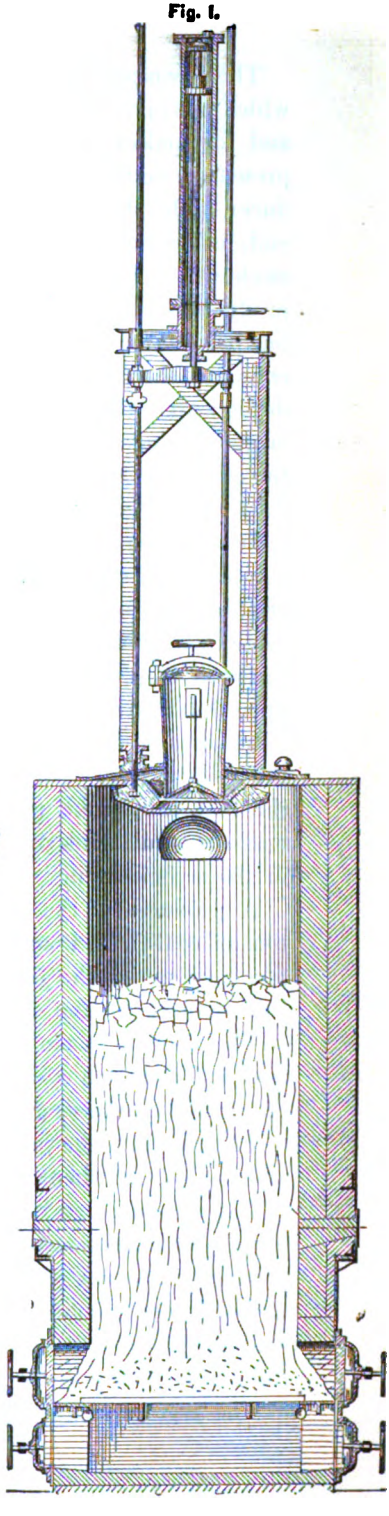


Fig. 1.



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The accompanying figures show a producer similar to the one used, which was specially designed for a large production of producer-gas, and for making a mixture of producer-gas and water-gas. Fig. 1 presents a vertical section, and Fig. 2 an outside view of the producer. The shell is 20 feet high, 9 feet 2 inches in exterior diameter, and, inside the lining, 6 feet in diameter. The lines of the producer, as shown in Fig. 1, are vertical. This is a good construction for a producer in which many different kinds of fuel are to be gasified; but it is not advisable to copy these lines for every fuel. Experience has shown that the lines to be given to a producer should depend upon the nature of the coal to be gasified therein. Having large cleaning- and ash-pit doors on both sides, opposite each other, the producer can be cleaned without great difficulty, even when a large accumulation of clinkers has taken place.

In order to get a gas very low in carbonic acid, a much larger depth of fuel than is generally carried in producers was required; and, as it is impossible to poke by hand such a deep fire, a pneumatic rammer was placed upon the producer. This rammer consists of a cast-iron ring, so constructed that it will not only exert a pressure upon the coal, but also force the coal to the periphery of the producer —which is desired, because the gas has a tendency to creep up along the walls. The ring is raised by air-pressure and allowed to fall upon the fuel, the stroke given depending upon the blow necessary to make the fuel sink regularly. The construction is clearly shown in the accompanying figures.

A producer of the above type can burn at least from 12 to 15 tons of coal in 24 hours. A positive air-blast is used, and steam is admitted under the grate, the admission of both being controlled by valves at the top of the producer.

The result of a ten days' run, carefully metered, showed that 1 ton of Pittsburgh coal will give, on an average, 131,280 cubic feet of purified producer-gas, corrected for barometer and temperature.

The gas from the producer passed through scrubbers, condensers and meters, and thence into a holder of 50,000 cubic feet capacity.

The carbonic acid in the gas was at times as low as 1.4 per cent., but the average percentage was about 3.4 per cent., at a pressure of 4 inches of water. With a pressure as low as is generally carried on producers in connection with steel-melting furnaces, the carbonic acid could have been easily kept at about 1 per cent. with such a deep body of fire in the producer.

The average composition of the gas was:

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CO_2	3.4
CH_4	3.1
H	9.2
C_2H_4	0.8
CO	25.3

The remainder was mainly nitrogen.

This experimental determination of the amount of producer-gas obtainable from 1 ton of Pittsburgh coal, will enable engineers to judge what they are likely to get from other coals.

An interesting fact as to the heat-units required to puddle 1 gross ton of pig-iron was determined by the engineers of the above company. The experimental gas-furnace employed, with regenerators for air only, was designed and superintended by Mr. W. F. Zimmermann.

The ordinary single-handed puddling-furnaces used in Pittsburgh consume about 36,000,000 British heat-units per gross ton of pig-iron, whereas, when gas is employed, about 14,000,000 heat-units will do the same work. It was found that 13,250 cubic feet of natural gas, with 1,100,000 heat-units per 1000 cubic feet, will puddle 1 gross ton of pig-iron, whereas, by using a gas with nearly 300,000 heat-units per 1000 cubic feet, 47,000 cubic feet were consumed, and by using a gas with 266,000 heat-units, 54,000 cubic feet were used to puddle 1 gross ton of pig-iron. The number of cubic feet of gas consumed, multiplied by the respective heat-units in a cubic foot, will, in each instance, give somewhat above 14,000,000 heat-units.

These results clearly show, that if all circumstances as to the production and consumption of heat could have been considered in these experiments, the amount of gas used in each case would have been exactly inversely proportional to the heat-units contained in the gas. This result could, of course, be expected from the laws of heat, but it is, nevertheless, an interesting fact to have it experimentally demonstrated on a large scale.

This result on puddling, and similar results on the consumption of different gases to evaporate water, combined with many other considerations, have led to the important conclusion that it is not advisable to distribute a gas, for domestic use, with a comparatively low amount of heat-units per 1000 cubic feet. The higher the heat-units per 1000 cubic feet, the better, especially when the gas is to be widely distributed.

I hope that the company which has kindly permitted me to pub-

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lish the above facts at this time, will soon find it to its interest to permit the publication of further facts relating to the interesting and complicated fuel-gas problem.

Discussion.

William H. Blauvelt, Philadelphia, Pa. (Communication to the Secretary): The presentation of this paper at the same meeting with that of Mr. W. J. Taylor on "The Energy of Fuel" is very fortunate; for, giving as it does, among other things, the results of direct experiments on the volume and composition of producer-gas from a ton of coal, we are enabled by it to examine more intelligently some of the theoretical statements in Mr. Taylor's paper.

It is specially interesting because it is the first time data have been published in which the pounds of coal gasified, and an analysis of the same, the number of cubic feet of gas made therefrom and its analysis, have all been given, so that we can calculate quite accurately the loss of energy in the gasification of bituminous coal, and, what is more interesting, determine just where the loss occurs.

To show this fully, I have taken the analysis by volume of the gas from a ton of coal, as given by Mr. Goetz, and, the total volume being given, have worked it back into cubic feet and pounds of gas, and these into pounds of carbon, hydrocarbon, hydrogen, etc. The result is tabulated as follows:

Gas from One Ton of Coal, Calculated at 62° F. and 760 m.m. Pressure.

Analysis by vol.	Per cent.	Cubic feet.	Pounds.	Equal to
CO	25.3	33,213.84	2451.20	1050.50 lbs. C + 1400.7 lbs. O.
H	9.2	12,077.76	63.56	63.56 " H.
CH_4	3.1	4,069.68	174.66	174.66 " CH_4.
C_2H_4	0.8	1,050.24	77.78	77.78 " C_2H_4.
CO_2	3.4	4,463.52	519.02	141.55 " C + 377.47 lbs. O.
N (by difference)	58.2	76,404 96	5659.63	7350.17 " Air.
	100.0	131,280.00	8945.85	

Calculated upon this basis, the 131,280 feet of gas from the ton of coal contained 20,311,162 H. U., or 155 H. U. per foot, or 2270 H. U. per pound.

I repeat here for convenience the composition of the coal from which this gas was made, as given by Mr. Goetz:

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	Per cent.
Water,	1.26
Volatile matter,	36.22
Fixed carbon,	57.98
Sulphur,	0.70
Ash,	3.78

One ton contains 1159.6 pounds carbon and 724.4 pounds volatile combustible, the energy of which is 31,302,200 H. U. Hence, in the processes of gasification and purification there was a loss of 35.2 per cent. of the energy of the coal. To illustrate this farther, I give below a theoretical gasification after Mr. Taylor's plan, of the coal from which this gas was made, and then compare the energy of the two gases.

Gasifying 1159.6 pounds carbon and 724.4 pounds volatile combustible (CH_4?), and deriving $\frac{3}{4}$ of the necessary oxygen from air, and $\frac{1}{4}$ from water, and oxidizing 10 per cent. C to CO_2, we have:

	Pounds.	Cubic feet.	Analysis by vol.
1043.64 pounds C to CO,	2435.16	32,996.42	27.2
115.96 " C to CO_2,	425.18	3656.54	3.0
724.40 " $VC-CH_4$(?),	724.40	16,893.00	13.9
O to gasify $\frac{1}{4}$ of above taken from H_2O liberates H,	53.25	10,117.50	8.3
O for $\frac{3}{4}$ of above from air leaves N,	4266.75	57,601.12	47.5
	7904.74	121,264.58	99.9

	Heat units.
Total energy in gas,	28,321,567
Energy per cubic foot,	1233
Energy per pound,	3583
Per cent. of energy of coal delivered in gas,	90

It must be borne in mind that the volatile hydrocarbons (which we call for convenience CH_4) are largely in the condition of vapor instead of fixed gas, and, as pointed out by Mr. Taylor, would not appear in the actual analysis; but this vapor is just as useful as if it were fixed gas, when not permitted to condense before reaching the combustion-chamber. Therefore, although the above gasification is theoretical, it probably represents approximately the real energy and composition of the gas which we would be able to deliver under favorable circumstances; and the difference between this analysis and the one first given shows that the complete utilization of the hydrocarbon vapors is of the highest importance.

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It is well known by chemists that when hydrocarbons are distilled at a high temperature they are in a measure broken up, the hydrogen being set free, and the carbon deposited in the form of soot. This would indicate that, in producer-practice on soft coal, we should aim to distil off the hydrocarbons in the top of the producer at a low temperature, just high enough to prevent the condensation of tar in the flues before reaching the furnace. In my experience, when the flues are short there is little danger of the latter occurrence, but even with a high fuel-bed, and consequently a cool producer-top, a small amount of soot is deposited in the flues.

In my opinion a high fuel-bed is as important for the reason above given, as for that mentioned by Mr. Goetz, viz., keeping the carbonic acid low in the gas, particularly when using fuels high in volatile combustible. There is no doubt that the practice of running soft-coal producers on a fuel-bed of from 24 to 30 inches, which has been so generally followed, is entirely wrong, both in theory and practice.

In my discussion of the gas-analysis given by Mr. Goetz, I have calculated the pounds of fixed carbon and volatile hydrocarbon that would be required to make this gas, and it may be interesting to look into this a little.

The composition of the hydrocarbons in a soft coal is uncertain and quite complex; but the ultimate analysis of the average coal shows that it approaches quite nearly to the composition of CH_4 (marsh gas), and I have considered it as such throughout my calculations, except that I have only allowed them 20,000 heat units per pound in calculating the energy.

Referring to my first table, 1192.05 pounds carbon and 252.44 pounds volatile hydrocarbons were required to form the gas of the analysis given, while in the coal from which it was made there were 1159.6 pounds carbon and 724.4 pounds volatile hydrocarbon. Hence, in the gasification and cooling process, 472 pounds, or 65 per cent. of the hydrocarbons, were lost. As these are so much the most valuable part of the fuel, the loss of 35.2 per cent. of the energy of the coal is easily understood. The gain of 35.45 pounds of solid carbon is no doubt due to the splitting up of some of the hydrocarbons in the producer, the carbon being oxidized to carbonic oxide or carbonic acid and the hydrogen set free.

The nitrogen in the gas-analysis (which is determined by difference) is probably a little high; but assuming it as correct, 95 per

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cent. of the oxygen necessary for the gasification of the carbon is derived from air, while 5 per cent. is derived from water.

This 5 per cent., or 88.9 pounds, liberated 11.1 pounds hydrogen. The 32.45 pounds carbon, referred to above as derived from the broken-down hydrocarbons, liberated 10.8 pounds hydrogen. This, with the hydrogen from the water, makes 21.9 pounds, leaving 41.66 pounds still to be accounted for. This 41.66 pounds is equivalent to 166.6 pounds marsh gas or hydrocarbons, containing 125 pounds carbon. Therefore, applying the theory of the breaking-down of the hydrocarbons, these 125 pounds carbon were deposited as soot in the flues and scrubber, while the remainder of the hydrocarbons, 294.6 pounds, were condensed as tar.

Mr. Taylor has pointed out in his paper the advantages of using as much steam in blowing a producer as will permit good incandescence to be maintained, and has shown that 25 per cent. of the oxygen necessary for combustion should be derived from steam without bringing the temperature in the producer too low.

Comparing the pounds of the various gases in the theoretical gasification of the coal under consideration with the respective quantities actually made, it will be seen that beside the difference in the volatile hydrocarbons, which has been already discussed, the great difference is in the nitrogen, there being 1400 pounds more in the actual than in the theoretical gas, thus diluting it enormously. This comparison between the gases, in which 5.3 per cent. and 25 per cent. respectively of the oxygen for the gasification of the carbon was derived from water, shows very clearly the advantages of the latter course. The fact that with plenty of steam much less clinker is formed in the producer is a very important one practically, though less definitely estimable in a theoretical discussion.

In the foregoing I have attempted to bring out the following points as highly important in soft-coal producer-practice:

First. That a large percentage of the energy of the coal is lost when the gas is made in the ordinary low producer and cooled to the temperature of the air before being used. To prevent these sources of loss, the producer should be placed so as to lose as little as possible of the sensible heat of the gas, and prevent condensation of the hydrocarbon vapors. A high fuel-bed should be carried, keeping the producer cool on top, thereby preventing the breaking-down of the hydrocarbons and the deposit of soot, as well as keeping the carbonic acid low.

Second. That a producer should be blown with as much steam mixed with the air as will maintain incandescence. This reduces the

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percentage of nitrogen and increases the hydrogen, thereby greatly enriching the gas. The temperature of the producer is kept down, diminishing the loss of heat by radiation through the walls, and in a large measure preventing clinkers.

Gas-producers promise to become a more important feature of our industrial works in the near future, and producer-gas must have the benefit of the same scientific study that is given to other forces, which, when intelligently directed and controlled, tend to greater economy and efficiency.

BIOGRAPHICAL NOTICE OF FRANKLIN B. GOWEN.

BY ECKLEY B. COXE, DRIFTON, PA.

(Washington Meeting, February, 1890.)

SINCE our last meeting, the Institute has lost, by the death of Mr. Gowen, one of its most distinguished members. I shall only attempt in this place to give a brief account of his many accomplishments and achievements; but our records would certainly be incomplete without a recognition of the important influence which he exerted upon the mining and metallurgical industries of this country, and a testimony to his private virtues of character.

Although Mr. Gowen was best known as a great lawyer and as the head of a great railway corporation, he began his career, before he was of age, as the manager of a small anthracite blast-furnace at Shamokin, Northumberland County, Pennsylvania. He left this position, to engage on his own account in the mining of anthracite coal at Mt. Laffee, near Pottsville. This proved an unprofitable business, and, after the panic of 1857, he began the study of the law. As soon as he was admitted to the bar, he at once entered upon the active practice of his profession, and soon achieved an extraordinary success. In 1869 he was made President of the Philadelphia and Reading Railroad Company, and he afterwards organized the Philadelphia and Reading Coal and Iron Company, the largest coal-mining corporation in the world. He became a member of this Institute during the first year after its organization, and remained a member until his death. He always took a deep interest in the work, and was prompt to co-operate by all means in his power in furthering the objects of the Institute.

Mr. Gowen was one of the first to recognize the great importance to both miner and consumer, of the proper utilization of the small

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sizes of anthracite which, for many years, were thrown away in the culm-banks. While at the head of the Coal and Iron Company, he did much to bring such materials into general use. At the meeting of the Institute held in Philadelphia in June, 1876, he delivered an address of welcome, which not only was, like all his speeches, eloquent and graceful, but also acquired a permanent professional interest from its forcible discussion of the subject of the utilization of anthracite waste and its lucid explanation of the system for burning coal-dust, invented by Mr. John E. Wootten, an officer of the Reading Company. As is well known, Mr. Gowen introduced this device in the locomotives of the railroad which he controlled, and it is still extensively employed in connection with both locomotive and stationary boilers.

Mr. Gowen was thoroughly convinced of the value of scientific training, and in hearty sympathy with every endeavor to elevate the professional standard of engineers. This was the secret of his interest in the Institute, which he held to be an effective agency in that direction. As an employer of engineers, he selected men worthy of confidence, and then trusted and supported them loyally. The testimony of all who ever served under him, will be found unanimous in this respect, and, on the other hand, the record made by the scientific employees of the Reading Coal and Iron Company, proves that the confidence of its head was not misplaced or abused.

But Mr. Gowen's greatest claim to the admiration and gratitude of his countrymen was due to the courage, perseverance and skill with which he pursued and destroyed the organization of assassins, known as the Molly Maguires, terminated the reign of terror they had maintained in the anthracite regions, and inaugurated in its place an era of peace, safety and order, which has not yet passed away. The full measure of his merit in that undertaking, cannot be appreciated, without considering that he vindicated law as well as justice. The condition of affairs had become so intolerable that many citizens of the better class were seriously considering the formation of a "vigilance committee," to take the law into its own hands, and protect by violence the life and property which violence had put in daily peril. Without stopping to consider in what case this extreme resort might be justified, we must admit that it is an evil, even when it is a necessary one, and that, after the achievement of the main ends sought through such measures, there must follow a period during which the confidence of the citizen in the regular forms of law is weakened, to be restored by slow degrees only. Mr.

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Gowen, when consulted in the matter, invariably and emphatically refused to take part in unlawful means of any kind, and maintained that the remedy should be, and could be, found in the regular proceedings of the established courts of justice. To the demonstration of this proposition he devoted his great abilities with a splendid audacity, pertinacity and acuteness, risking his life freely, and knowing neither discouragement nor rest until he had brought to trial, conviction and execution, the chiefs and agents of the conspiracy. The moral effect of his campaign was worth that of a hundred vigilance committees. It is gratefully recognized to-day in thousands of peaceful homes throughout the coal-mining districts of Eastern Pennsylvania, and, indeed, I should scarcely be guilty of exaggeration if I declared that the example set by Franklin B. Gowen, in the suppression of anarchy and crime by the ordinary methods of civilized government, has been an example to the friends of order and justice throughout the world.

Of his personal character, I hardly dare trust myself to speak. For nearly a quarter of a century, he was one of my closest and dearest friends, as well as my trusted legal adviser in all my complicated business relations. He was the soul of honor, a gentleman of the highest and best type—possessed of a courage both moral and physical, that I have never seen equalled. I esteem it one of the greatest privileges of my life to have known so intimately a man of such a high sense of duty, and one so truly appreciative of the true distinction between right and wrong. Against opponents who were strong and well able to take care of themselves, he was an uncompromising adversary; but towards the poor and weak he had a tenderness almost womanly. Of his innumerable kindnesses to the unfortunate, none save himself ever knew more than a small part; but many have come to my knowledge in various indirect ways.

His love for home and family was deep and strong, and his private life was spotless.

Born February 9, 1836, and dying December 14, 1889, he compressed into a life that scarcely passed its prime a wonderful record of persistent activity. At a comparatively recent period, he had resigned his presidency of the Reading Company, and had resumed the practice of his profession as a lawyer, with every prospect of repeating and surpassing the brilliant successes of earlier years. The tragedy of his premature death has inflicted upon his friends and his country an unspeakable loss.

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BIOGRAPHICAL NOTICE OF WILLIAM R. JONES.

BY R. W. RAYMOND, NEW YORK CITY.

(Ottawa Meeting, October, 1889.)

THE sudden death of Captain Jones, September 28, 1889, became known to his fellow-members on the eve of the Ottawa meeting of the Institute; and at that meeting, Mr. Charles Albert Ashburner, of Pittsburgh, expressed with unaffected feeling and unpremeditated eloquence the sorrow of all. Mr. Ashburner subsequently undertook to revise and extend, for publication in the *Transactions*, the remarks he had made at Ottawa; but death surprised him also, in the midst of his labors; and it has become my duty and privilege to prepare from the memoranda he left behind, the present brief and inadequate notice. To me, who knew and loved them both, this task of double sorrow recalls them both, with a peculiar sense of presence and of absence. They seem to speak from another land, not far away, yet not with human voices—like an echo and its echo.

William Richard Jones was born in Luzerne county, Pennsylvania, February 23, 1839. His father was Rev. John G. Jones, a Welsh minister, who had immigrated to the United States in 1832. William was the eldest son, and his father's failing health made it necessary for him to begin early the work of self-support. At the age of ten, he was apprenticed to the Crane Iron company, of Catasauqua, and in this humble capacity entered upon the profession in which he was to become famous. Unquestionably, the thorough acquaintance with all details, acquired by working his way from the bottom to the top, was one secret of the conspicuous executive ability which characterized him in the height of his career.

According to a letter from Mr. J. H. Geer, Assistant Chief Engineer of the Cambria Iron Company, Mr. Jones entered the service of that company in 1858 or 1859 as a machinist, and continued to work in that capacity until July, 1862, when he enlisted as a private in the 133d Regiment of Pennsylvania Volunteers, in which he afterwards commanded a company. The regiment was mustered out of service in 1863, and Captain Jones re-enlisted in the 194th

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Pennsylvania. After his final discharge, he resumed his connection with the Cambria Iron Company, and became assistant to George Fritz, the brilliant mechanical engineer, then in charge of its works at Johnstown. Until Mr. Fritz's death, Captain Jones was busy under his directions in designing, erecting and perfecting the great Bessemer blooming-mill, and other plants of the company.

This position he left to take the place, first of master mechanic, and then of general superintendent, of the Edgar Thomson Steel Works at Braddock, near Pittsburgh. I believe his selection for this trust was due in no small degree to the admiration and confidence of his staunch friend, Alexander L. Holley, who, as designing and consulting engineer of all the great Bessemer works, was brought into contact with the ablest men in that department, and was accustomed to rate Captain Jones as one of the best practical administrators among them.

It was under Captain Jones's direction that the famous blast-furnaces, A, B, C, D, E, F and G, of the Edgar Thomson plant, were erected. As is well-known, an outburst of molten iron and cinder from one of these furnaces inflicted the injuries of which, after lingering two days, he died.

Like all mechanical engineers engaged in the daily routine of administration, Captain Jones devised and executed many things, as the ordinary incidents of duty, for which professional inventors would have taken patents, and it is rather in the works he constructed or re-constructed than in any official record of inventions that the evidence of his fertile ingenuity is to be sought. Yet the number and importance of the patents issued to him is not insignificant, as the following list, for the completeness of which I cannot vouch, may testify:

1876. Device for operating Bessemer ladles.
1876. Improvements in hose-couplings.
1876. Fastenings for Bessemer converters.
1876. Washers for ingot-moulds.
1877. Hot-beds for blending rails.
1878. Apparatus for compressing ingots.
1881. Cooling roll-journals and shafts.
1886. Feeding appliance for rolling-mills.
1886. Manufacture of railroad-bars.
1888. Apparatus for removing and setting rolls.
1888. Housings for rolls.
1889. Apparatus for removing ingots from moulds.

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1889. Method for mixing in receiving-tanks the metal from blast-furnaces.

For the last invention, which, I believe, he considered to be the most important in this catalogue, the patent had been allowed, but not issued, at the time of his death.

Captain Jones joined the Institute in 1875, and was always deeply interested in its progress and success, although he persistently declined to accept nomination for office, and (except in the way of discussion) contributed no papers to the *Transactions*. His assistance and hospitality to his fellow-members were ready and cordial.

He was also a member of the American Society of Mechanical Engineers, the Engineers' Society of Western Pennsylvania, and the Iron and Steel Institute of Great Britain. A daughter and son survive him. The latter, Mr. W. M. C. Jones, is now employed as an engineer in the Edgar Thomson Works.

Of the personal qualities, which made Captain Jones universally esteemed and beloved, I need say nothing to those who knew him. The mere mention of his name calls up his stalwart form and jovial face—the picture of frankness, humor, courage and energy. As a manager of men, he combined justice and sympathy with uncompromising authority. Perhaps, no single scene of his career was more picturesquely characteristic than the arrival at Johnstown, after the great catastrophe of the flood, of "Captain Bill Jones," with the men of Braddock—a host of willing workers, with a leader who was himself a host—to inspire hope, maintain order, and set up in the midst of confusion and paralysis, a model of hearty, well-directed, disciplined and effective activity.

I cannot more fitly close this imperfect tribute than with the testimony of the members of Carnegie, Phipps & Co., Limited, to their trusted and beloved consulting engineer.

At a joint meeting of the managers of Carnegie Brothers & Co., Limited, and of Carnegie, Phipps & Co., Limited, held at their branch offices, 48 Fifth avenue, Pittsburgh, Pa., on October 3d, 1889, the following resolutions were adopted:

Resolved, That, as this firm, in all its history, has never been called upon to record a loss so tragic as that which has deprived it of its great manager, CAPTAIN WILLIAM RICHARD JONES, so neither has it ever lost an officer whose services were more valuable or to whom it was more deeply indebted for the success which has attended its operations.

Resolved, That, invaluable as the services of Captain Jones have been, not only to us but to steel manufacture in general, the remembrance of these fade away in the keen pangs of grief awakened in us by recollections of our friend, the man.

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Resolved, That the history of steel manufacture will record his name with those whose joint labors have brought the art to its present stage of perfection, and in the list of men who have risen from the ranks through the possession of indisputable genius to commanding positions as organizers and managers of masses of men in the industrial armies of this age, the highest rank must be accorded to Captain Jones.

Resolved, That, to us, his employers and friends, who knew him intimately through many years of almost daily intercourse, there is still left in our grief, though he has gone, the precious privilege of meditating upon a combination of manly qualities, which constitute the real man, and which, united in him, gave forth that indefinable but rarest quality, character; a brave, just, honest, transparent soul; a staunch, loyal-hearted, generous friend was he, whose absence from us and from our counsels we to-day so deeply mourn.

Resolved, That his life should have been sacrificed in our service must ever hereafter tinge our thoughts of the Edgar Thomson Steel Works with feelings of profound regret, and yet we would not forget that the Commander fell at the head of his men, at the post of duty, amid the roar of the vast establishment, which was his work, and which is his monument. A heroic end was his, worthy of the soldier he was; for Captain Jones fell upon the field which he had conquered.

Resolved, That the firm cannot forget the faithful men who died or suffered with their Captain. It begs to thank the survivors and to assure them that it appreciates, to the fullest extent, their devotion to duty; and the profound sympathy of the firm is hereby offered to Mrs. Jones and her family, and to the families of those who were stricken by the recent grievous accident.

Resolved, That these resolutions be engrossed upon the minutes and copies forwarded to the bereaved families.

RAILWAY SPLICE-BARS AND SPECIFICATIONS FOR THEIR MANUFACTURE.

BY ROBERT W. HUNT, CHICAGO, ILL.

(Washington Meeting, February, 1890.)

No matter how perfectly sectioned or skilfully manufactured the rails used may have been, or with what care the road-bed may be maintained, legitimate results from these important factors cannot be secured unless proper attention is bestowed upon the joint-fastenings.

In this country, splice-bars of various patterns furnish the practically universal joint. Other forms have been used, and are now being experimentally tested, but the typical American rail-joint consists of angle-plates or splice-bars. At the same time, it is true that scarcely a railroad engineer is satisfied with his joint. He may consider it the best of which he has knowledge, but at the same time he is well aware of many of its weak points.

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Even after the discovery of the much-desired and long-sought "perfect-joint," it will require some years to displace and replace the many thousand tons of angle-plates now in service. But, apparently, this discovery has not yet been made and, while we are waiting for it, it would be well to endeavor to secure as good a splice-bar as is practicable under existing commercial circumstances.

I do not propose discussing sections, as they must depend so much upon the rail used, character of joint, etc., but will confine myself to presenting specifications for manufacture which will, in my judgment, help engineers to obtain well-finished bars of good material, and which, at the same time, ought not to add anything to their cost.

As is well known, practically, all iron splice-bars are at present made from old iron rails, or iron scrap. The supply of the former material is rapidly diminishing, and the quality of the latter is usually unknown and doubtful.

For these reasons, and because I believe the metal will in every way make a better bar, I favor steel. I am fully aware that, in many cases, the use of steel splices has not been satisfactory; but I believe these failures resulted from faults either in the grade of steel used, or in the physical treatment of the bars.

Steel rails, steel bridge-members and steel boiler-plates were all condemned in their early history, but are now conquerors in their various fields. In my specifications I have endeavored to provide for suitable material, whether it be either iron or steel, and I insist upon a practically perfectly-finished bar. If bars are not straight in surface, they cannot fit the "fish-section" of the rails and, if out in line, they must either not come to place, or the track-bolts must perform service for which they are not designed. If, in punching, the bars are bulged opposite the bolt-holes on either top or bottom-line, it is evident that they will bear against the upper side of the rail-flange, or the under side of the rail-head at those points only. Under all circumstances, the punching and slotting should be carefully done; but this is of prime importance when steel is used. I, therefore, call for the use of the Kennedy spiral, or some other shearing-punch. Such tools put considerably less strain upon the metal, and are by so much easier upon the punching-presses. Hence the interests of both producer and consumer will be well served by adopting them.

It is necessary that the slots in steel plates should have well-defined fillets at their corners.

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With these introductory remarks, I submit my proposed specifications.

Specifications for Iron Angle-Splices.

Section 1.—The iron shall be of good quality, and plates taken from among those offered for acceptance, when nicked on one side and broken suddenly over an anvil, at a freezing temperature, must show at least 70 per cent. of fiber. If four plates out of six, taken at random from a car-load, fail under this test, the whole car-load shall be rejected.

Sec. 2.—The bars shall be rolled true to the templates represented by the blue-prints which will be furnished to the rolling-mills by the railroad company. They shall be rolled smooth, and without fins or checks, and punched and slotted accurately to the dimensions given in the blue-prints, and specified in the orders or contracts.

Sec. 3.—The punching shall be performed in such a manner that the plates will not be bulged on either the upper or lower lines opposite the bolt-holes, and the holes must be free from burrs. The Kennedy spiral, or other shearing-punches, shall be used. The ends of the plates shall be without burrs or kinks, and the plates shall be practically straight in all directions.

Sec. 4.—The railroad company's inspectors shall have power to reject any plates in the manufacture of which *any* of the above specifications has been violated.

Specifications for Steel Angle-Splices.

Section 1.—The metal shall be made by either the pneumatic or the open-hearth process.

Sec. 2.—By preference, it shall not contain over 0.10 per cent. of phosphorus and not over 0.17 per cent. of carbon. Should the phosphorus exceed 0.10 per cent., the carbon shall not be over 0.12 per cent.; but the phosphorus shall never exceed 0.12 per cent.

Sec. 3.—The steel must be heated with care, and in no case shall the heat be brought up to the limit of safety.

Sec. 4.—The bars shall be rolled true to the templates represented by the blue-prints which will be furnished to the rolling-mills by the railroad company. They shall be rolled smooth, and without fins or checks, and punched and slotted accurately to the dimensions given in the blue-prints, and specified in the orders or contracts.

Sec. 5.—The punching shall be performed in such a manner that the plates will not be bulged on either the upper or lower lines opposite the bolt-holes, and the holes must be free from burrs. The Kennedy spiral, or other shearing-punches, shall be used. The ends of the plates shall be without burrs or kinks, and the plates shall be practically straight in all directions.

Sec. 6.—Rolling-mill companies, furnishing plates under these specifications, shall give to the inspectors analyses showing the amount of carbon in all the heats of metal which are used in making the plates and, whenever required, shall also give the amount of phosphorus.

Sec. 7.—The railroad company's inspectors shall have power to reject any plates in the manufacture of which *any* of the above specifications has been violated.

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STRIPPING ORE-DEPOSITS.

BY F. H. McDOWELL, NEW YORK CITY.

(Washington Meeting, February, 1890.)

THE shipment during 1889 of 68,221 tons of ore from the Tilly Foster iron-mine represents the measure of success which has attended the pioneer enterprise in this country in the employment of stripping, as a scientific method and upon an immense scale, for the recovery of abandoned ore-reserves. A still further reduction in the cost of the work, below the figures given in my former paper (*Trans.*, xvii., 758), shows how cheaply such operations can be conducted under a carefully matured plan, supported by the judicious selection and use of modern appliances.

At a depth of 250 feet, the new slope given to the hanging-wall has been well maintained and the recovery of over 500,000 tons of ore as a minimum has been already assured. The intrinsic merits of this system having thus been established under the direction of Mr. E. S. Moffat, the General Manager of this Company, numerous owners of mineral lands are studying its advantages. The writer's association with the work at the Tilly Foster has led to frequent consultations with the managers of other companies looking to the application of the same system to the development of their deposits. Some interesting examples are illustrated in the present paper.

THE PETERS MINE, RINGWOOD, N. J.

About a year ago Dr. Raymond, as engineer of Messrs. Cooper, Hewitt & Co., together with Mr. Rothwell, was considering the removal of the ore-floors and pillars from the Peters Mine at Ringwood, N. J., where the limit of profitable working by the old method of mining had been very nearly reached. Mr. Ferdinand Ruttmann was engaged to make the survey which forms the basis of the accompanying diagrams. At first it was thought that some method of mining and filling might be introduced with advantage; but after a careful study of the operations at the Tilly Foster it was decided to adopt the system of stripping.

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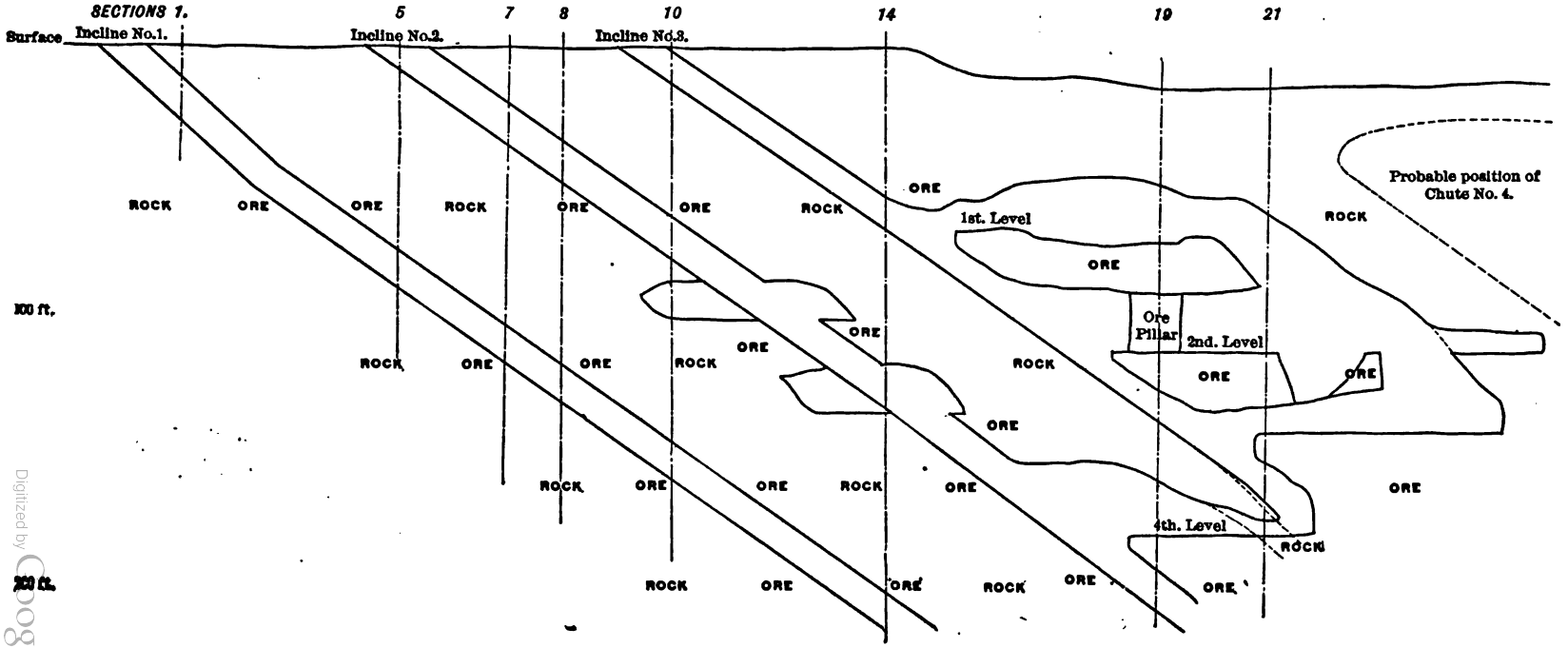
FIG. 1.



Longitudinal Section of Peters Mine, Ringwood, N. J.

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By reference to the longitudinal and cross-sections shown in Figs. 1, 2 and 3 it will be seen that while the Peters vein stands nearly vertical the ore-shutes overlie each other at an angle of 35° and from 10 to 30 feet apart. It is proposed to take advantage of this and to pile up the waste on the end-wall of the lowest chute as rapidly as the wall is uncovered through the removal of the ore.

This system of back-filling will not only make the mine much safer, as the new hanging-wall slopes will be supported almost as soon as formed, but much excavation will be saved by making the slopes considerably steeper than if the pit were to be left open permanently. To insure rapid and economical handling of the material, Mr. E. G. Spilsbury has advised the introduction of steel wire-rope cables, such as have been used with signal success at Tilly Foster, but suspended lengthwise over the vein instead of at right angles to it. A large body of ore has already been uncovered, and, depending upon the depth ultimately reached, from 100,000 to 200,000 tons will be recovered, and at a considerable saving in cost when compared with expenses incurred under the old system.

For the present, while stripping is confined to shallow depths, steam-derricks are used; but as sinking progresses the cable will be introduced. This will deliver all the waste to the end-wall and the ore to cars on the surface, the back-filling following the mining as closely as can be done conveniently.

There will naturally be a limit to the profitable mining by this system of each chute; and considerable judgment will be required in locating the final lines of deepest workings, since, owing to the inclination of the ore-deposits and the character of the hanging-wall rock, they will vary at every point along the line of the vein. While these limits, therefore, cannot be definitely fixed until the hanging-wall has been carefully tested as sinking progresses, the lines I have drawn in the accompanying diagrams, Figs. 2 and 3, will serve to make the subject clearer. At Section 1, which is taken at right angles across the vein 20 feet horizontally from the head of incline No. 1 we note that the earth to be excavated calls for a comparatively flat slope-line. At Section 5, which is taken 100 feet from the head of incline No. 1, chute No. 1 has dropped down to a depth of about 90 feet; chute and incline No. 2 have come into view; the amount of excavation has largely increased, and the slope-lines have been steepened, to correspond with the improved character of the material encountered.

Section 7, which is 140 feet from the head of incline No. 1, shows

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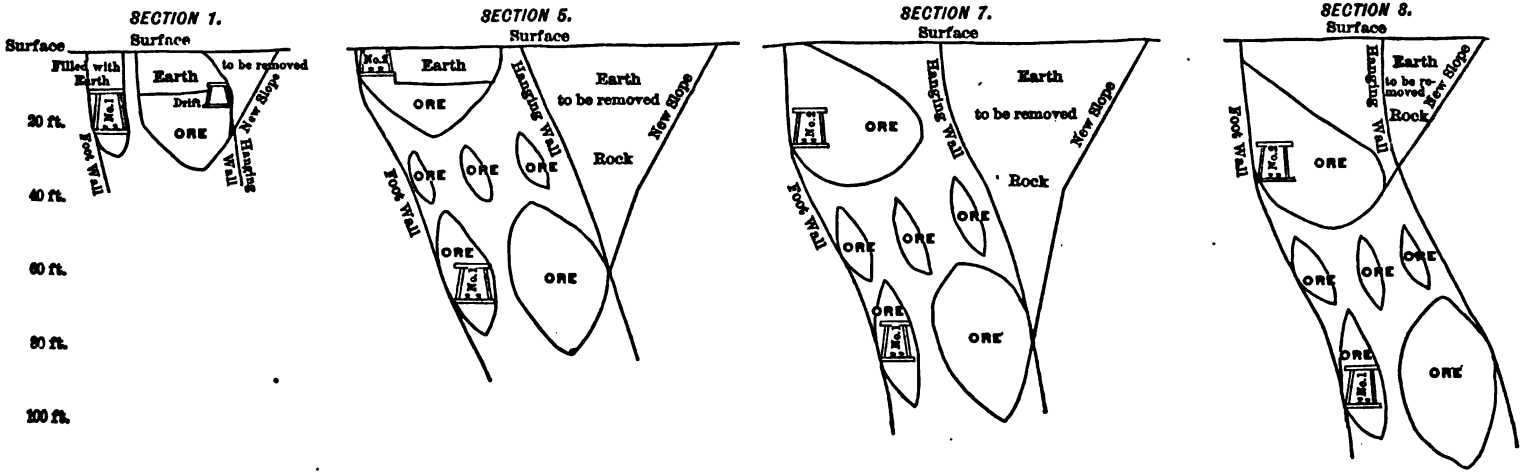
FIG. 2.



Cross-Sections of Peters Mine, Ringwood, N. J. (See Fig. 1.)

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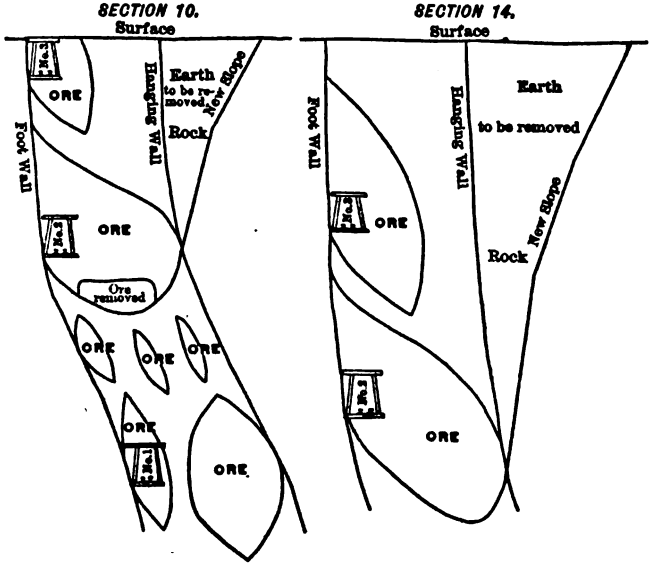
the probable limit of profitable workings on chute No. 1. The flattening of the vein makes it necessary to carry the excavation about 60 feet into the hanging-wall, the lower end of the wedge being 100 feet from the surface.

At Section 8 we are forced to abandon operations on chute No. 1 and confine them to chute No. 2.

At Section 10 all three chutes are shown with their respective inclines, although but two of these can be profitably worked.

At Section 14 is shown the greatest depth to which the workings in chute No. 2 can be profitably carried.

FIG. 3.



Cross-Sections of Peters Mine, Ringwood, N. J. (See Fig. 1.)

At Section 19 Fig. 4, we find it necessary to confine our operations to chute No. 3, unless, as is quite likely, a new overlying chute, which we may call No. 4, should be discovered.

While this section shows considerable "ore removed," probably an equal amount of ore will be recovered.

At Section 21, Fig. 4, we reach the probable limit of profitable workings on chute No. 3; and further stripping-operations to the north will depend upon the volume of ore discovered in the probable chute No. 4 and a possible No. 5.

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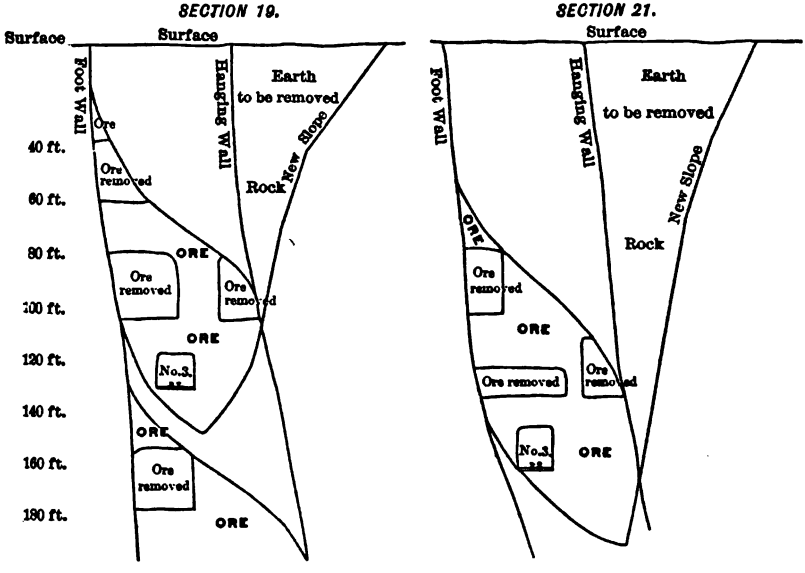
The Bertha Zinc-Mines, Va.

Another stripping-operation of considerable magnitude which is well advanced at Bertha, Va., under the direction of Mr. W. H. Case, Superintendent of the Bertha zinc-mines, is shown in Fig. 5. Here some 30 cubic yards of earth have been removed for every ton of ore mined. Underlying the earth-covering and overlying a series of irregular limestone points and the depressions between them, are found the zinc carbonates to the extent of about 8000 tons per acre. The average depth of earth is about 81 feet, making 130,680 cubic yards per acre, and the mining of a year's

Fig. 4.



Cross-Sections of Peters Mine, Ringwood, N. J. (See Fig. 1.)

supply of ore involves the movement of about 500,000 cubic yards of earth. As Mr. Case agrees with the writer in deeming this amount excessive, he has instituted a series of underground explorations looking to the ultimate exhaustion of the ore-deposits under the deepest earth-coverings by the old system of robbing through drifts, sinks and upraises. These will undoubtedly prove worthy of a special paper from Mr. Case, as he will shortly be in possession of sufficient data to make between the two methods of mining a comparison which will have much value for those who may be called upon to exploit similar deposits.

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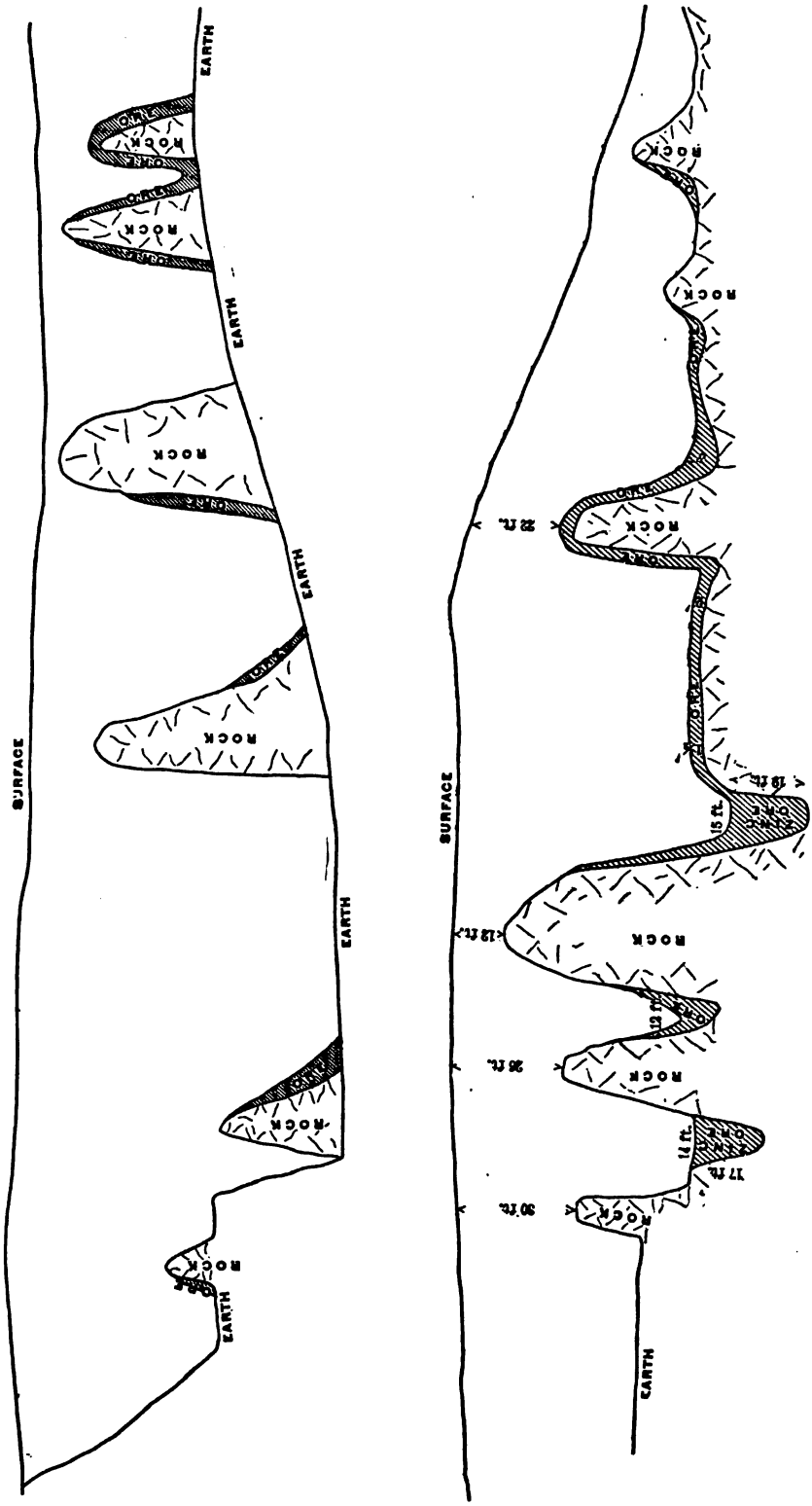
FIG. 5.



Vertical Longitudinal Section of Bertha Zinc-Mines, Virginia.
(The lower half is the right-hand continuation of the upper half.)

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As mining underground has not advanced far enough at present to insure ample furnace-supplies, Mr. Case has recommended that certain shallow deposits be uncovered by means of steam-shovels and suspended steel wire-cables, the former working off the surface-earth down to the limestone points; the latter completing the earth-excavation and removing the ore; the earth being moved back to the exhausted bottoms, and the ore delivered to cars for shipment.

It is quite likely that a similar combination of the two systems can be employed advantageously in connection with the mining of large deposits of both iron-ore and zinc-ore in the South. Probably the most extended deposit of this character is that now being opened by the New York and Virginia Mining and Mineral Company, of Virginia, at a number of points in Carroll county, Virginia, along the line of a great mineral vein or zone, carrying in depth pyrrhotite and chalcopyrite. Miles of drifts have been run by previous owners in their search for copper. These serve to show the immense quantity of gossan (iron-ore) in place, along the twelve miles in length of vein, ready for extraction.

As these deposits vary in depth from one foot to three hundred feet, and the stripping will be very light in some places, it will pay to establish open-pit workings at such points, and load directly into cars for shipment; at others it will be advisable to conduct operations underground, loading into narrow-gauge cars at the foot of the slope, and reloading from these into standard-gauge cars.

As mining can be pushed at numerous points at the same time, there will be no difficulty in handling one thousand tons of ore daily.

The Dannemora Mine, Sweden.

The members of the Institute will be interested to know to what depths open mining, after stripping, has been carried in Europe.

By reference to the accompanying longitudinal and cross-sections of the Dannemora Mine, of Sweden, shown in Figs. 6 and 7, which I have secured through the courtesy of Mr. Robert A. Cook, of New Brunswick, New Jersey, it will be seen that these open-workings have been carried to a depth of about five hundred feet. Below this point it has been deemed safest to conduct operations underground, and this has been made possible by the sinking of a new shaft outside the vein, in the undisturbed country-rock. In Fig. 6 the vein above the 160-meter line has been excavated and removed by open-work, except some pillars and "horses" of limestone, etc.

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Below that line it has been laid out for underground stopes in the usual way. The new shaft is shown in both sections. The ore mined is now replaced with a back-filling of waste mine-rock, some of which is lowered from the surface. Mr. Cook informs me that the open-workings of the copper mines at Fahlun, Sweden, have been carried to even greater depths.

Discussion.

R. W. Raymond, New York City: The successful application of stripping, as a systematic method of exploitation, although it presents no features of startling novelty, entitling it to be ranked among great inventions, is, nevertheless, in my judgment, one of the most important improvements which the art of mining has received during the present generation; and it is, like many other great improvements, a return to the simpler methods of the past, with the perfected means of the present. People have always mined in open pits when they could; but the increasing cost of the removal of waste has limited the extent of such workings; and moreover, mining-operations have been, in many cases, so unskilfully conducted by open-work as to render impracticable, unsafe or ruinously expensive the continuance underground of developments thus commenced. Hence, the method has scarcely been ranked as scientific mining. And above all, its application to solid rock-masses was scarcely dreamed of. Ore-banks and quarries might be dug out and blasted out; but not immense hanging-walls. As soon as an ore-deposit was found to dip into the solid country-rock we all agreed that it must be worked in the regular way, with shafts, levels, winzes, stopes, etc., and the hanging-wall must be held up, temporarily or permanently, by means of ore-pillars and floors left standing, or timbering or masonry, or "gob" and packing.

Now, no new principle has been discovered. The question was always merely one of cost. But it has been put in a new light by the combined effect of inventions and improvements which have reduced the cost of excavating and handling rock, and have, at the same time, permitted closer calculations to be made beforehand as to the required amount of such dead-work, and the time and money which must be expended upon it. I may mention as the chief of these improvements, the following:

1. The diamond-drill enables us, in many cases, to determine the position and extent of the masses which are to be exploited. In the case of the Tilly Foster mine, the requisite precise knowledge

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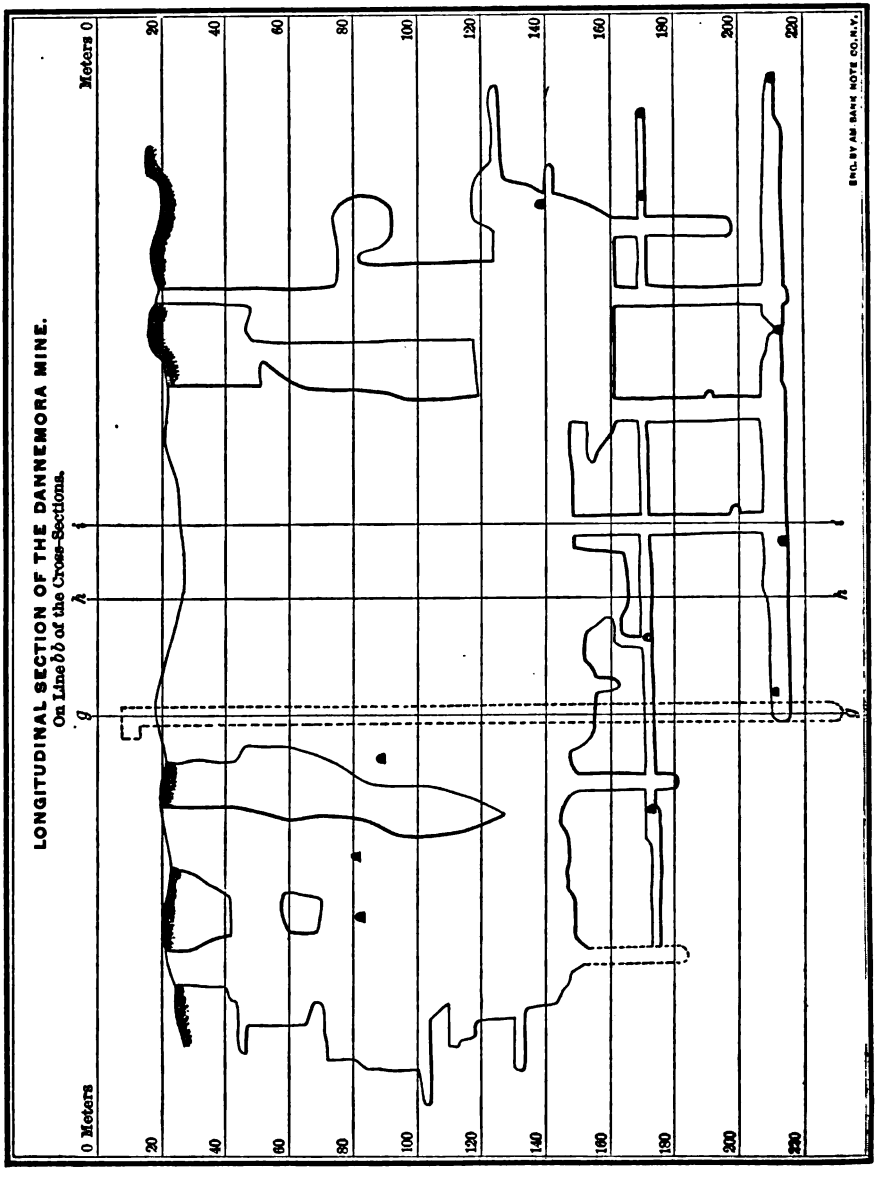
was obtained by actual operations underground, which developed on all sides the boundaries of the ore-body, and thoroughly revealed its average quality.

Fig. 6.



2. Power-drills have greatly reduced the cost of drill-holes for blasting; by far the greatest reduction under this head has been

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realized under the conditions of open-work, which permit the heaviest blasts.

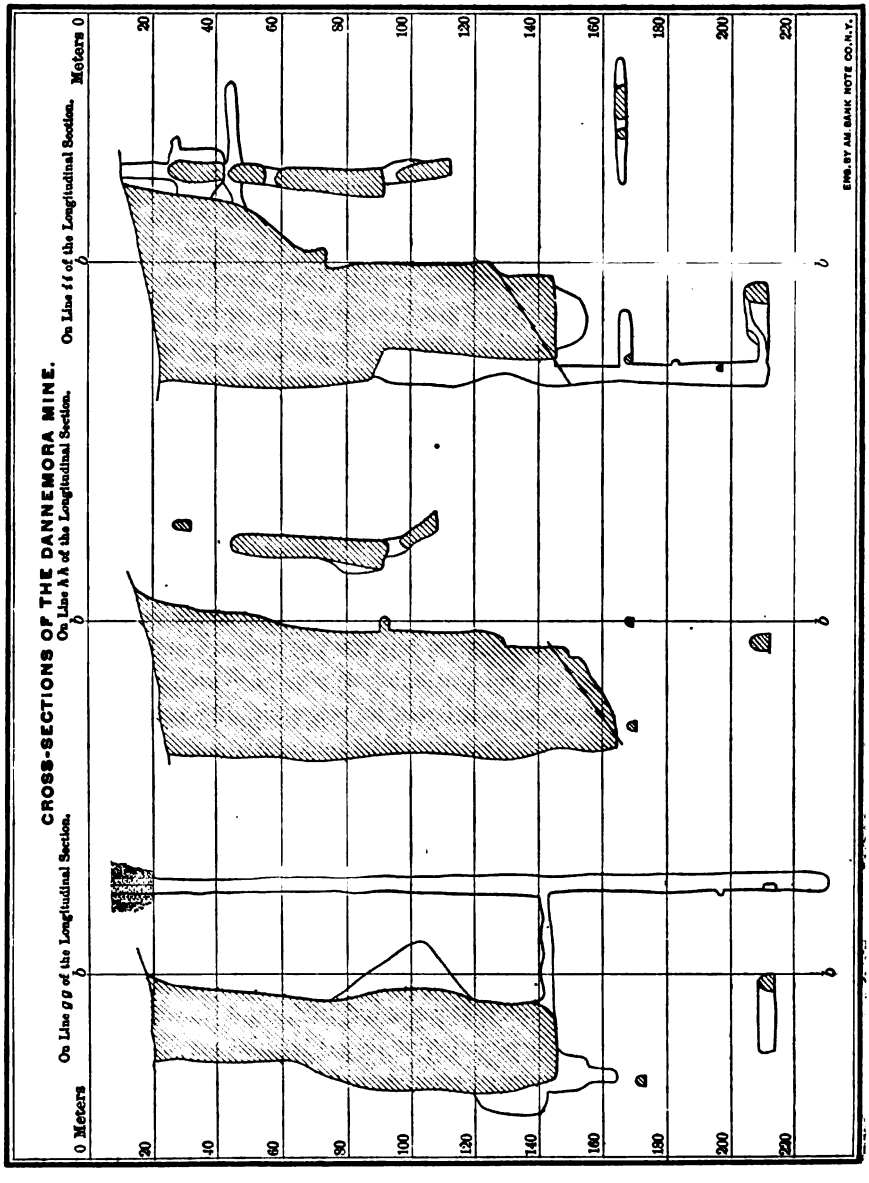

FIG. 7.

3. The same is true of the use of the various modern high explosives. While these have, to some extent, exhibited their best

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effects in small blasts and in confined underground spaces, yet, on the whole, it is in open-work that the skilful use of the power-drill, coupled with the skilful choice and employment of the proper explosives, has found the widest field for economic triumphs.

4. To these improvements must be added the introduction of mechanical methods for moving large quantities of material.

The gradual perfection of all these instrumentalities has been accompanied with the training of a generation of engineers competent to handle them, and of a generation of capitalists who have confidence in the engineers. I remember well the day when Mr. Moffat called at my office to talk over with me his plan of removing bodily two or three hundred thousand cubic yards of the gneiss hanging-wall of the Tilly Foster, at the cost of about as many dollars. It did not take him long to convince me that the thing was practicable; but it was hard for me to realize that his employers were willing to spend so much money in a lump on the strength of his calculations, and that responsible contractors would undertake and guarantee the job.

The operations at the Peters mine, belonging to the house with which I am connected as engineer, are neither so stupendous nor so definitely certain in result as those at the Tilly Foster. We are re-opening old ground, in which we encounter, from time to time, not merely our own former workings, but the burrowings of the ancient London company, of pre-revolutionary times, of which we have no maps. How much ore we shall win by our stripping remains to be seen. So far, I think I may say, we have come upon as many unexpected ore-bodies as unexpected holes. I refer to such relatively small outlying masses as are likely to be overlooked in underground work, particularly in our New Jersey magnetite mines. It has happened in my experience more than once, that what was supposed to be the true wall-rock of a vein has proved, on piercing it, to be merely a shell, behind which there was another layer of ore. More frequently still, however, it has happened that we have run into the wall-rock, hoping for such a piece of luck, and found nothing. Mine-managers know very well that the aggregate expense of these blind gropings in the country-rock, lest something of value should escape us, is a serious item in the expense of underground mining. The advantage of tearing down a hanging-wall in broad daylight, taking everything as it comes, and feeling sure that we have got it all, is no small satisfaction, and deserves to be considered as one of the favorable features of the stripping system.

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Finally, I desire to point out in the diagrams of the Peters mine and to prove by the sections of the Dannemora mine, as presented in Mr. McDowell's interesting paper, that there is nothing to prevent the employment of stripping to any convenient depth, and the continuance of the work, below that depth, by underground operations. This contingency must be kept in view, and nothing must be done that would make such a transition impossible; that is all.

GOLD-QUARTZ.

BY W. M. COURTIS, DETROIT, MICH.

(Ottawa Meeting, October, 1889.)

THERE seems to be a well-established belief that there is an indescribable something in the appearance of gold-quartz not to be learned from books, that enables those who are fortunate enough to have acquired the power to recognize this characteristic, to pronounce at once that a certain quartz will or will not pay. The stronger this claim is made, the more certain is the claimant to be one of the self-styled practical miners.

After several years spent among the gold-mines of California with practical and professional gold-miners, I have come to the conclusion that very few care to risk an opinion on a piece of quartz until some of it has been "horned out." Nevertheless, there is what the miners call a "kindly" appearance, that is more often followed by good results than is a vitreous, water-wet-looking quartz. A dark, resinous quartz seems to be most in favor, but there are so many exceptions that even quartz as "white as a hound's tooth" produces gold in some mines.

The ribbon-quartz, a laminated variety, separated by thin talc or chlorite partings, has a well-known good name. The slides of rich quartz specimens show that this parallelism is a microscopical characteristic of the only macroscopical distinction that, as a rule, these quartzes show. This is shown by Nos. 12 and 13, photographs of the richest quartz, that averaged $65 per pound. It is noticeable, also, in Nos. 7 and 9, and others.

Dr. Hensoldt, of Columbia College, who prepared, excellently, a large number of my slides, and whom I have to thank for descriptions and suggestions, says of Nos. 12 and 13: "This variety of quartz differs considerably, microscopically, from the others of your locality, and is, in more than one respect, remarkable. Like the

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others, it is crowded with fluid enclosures; but the cavities, instead of being irregularly scattered, are here arranged in parallel lines or rows with considerable symmetry. Many, if not all, of them contain vacuoles in a state of continual motion, and I am inclined to believe that the liquid they imprison is carbonic acid. When warmed and tested with a Stage thermometer, the bubbles suddenly disappear when a temperature of about 30° C. is reached, but return again on cooling.

"Now, between 30° and 31° is the so-called 'critical point' of liquified carbonic acid; that is, above this temperature carbon dioxide can no longer maintain its liquid condition, but transforms itself into a gas, no matter what the pressure may be. (This has been conclusively shown by Prof. Andrews, of Belfast, who thus determined the critical point of every known liquid.) I am not aware of any other liquid of which the critical temperature is so low.

"The inference to be drawn from this is, that inasmuch as carbon dioxide is a gas which can only be reduced to the liquid condition by extreme pressure, the quartz in which these cavities occur has been exposed to great pressure (tons to the square inch) during its formation or crystallization."

This parallelism is found more or less developed in most of the specimens of rich quartz on some part of the slide.

This quartz (Nos. 12 and 13), referred to by Dr. Hensoldt, occurred in small leaders coming obliquely from the hanging-wall into a large vein of block-quartz of very low grade. The appearance of the latter quartz suggested a second opening of the fissure after it was solid or nearly so. It is brecciated on the edges, and broken up into large blocks, which seem afterwards to have been cemented together with iron oxide and jasper, carrying some gold.

The remarks of the late Prof. Clayton on the quartz of the Drum Lummon lode, are fully confirmed in the character of this quartz in the Tiger mine. The fissure occurs in sericite schist, broken through by diorite dykes, and the quartz is accompanied by a parallel diabase band that sometimes entirely crowds out the quartz. Both diabase and diorite have been differently named from slides taken at various points. The limestones have been altered to marble; some of the dykes have been turned to soapstone. The country seems to be one of intense metamorphic action.

While millions of dollars have been taken from the surface of this district in the early placer days, and all the little gulches leading up to the veins have proved rich; yet, although pockets of rich

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quartz of considerable importance have been found, no one has made a success of working the veins. Many companies have been wrecked on them. The average value of the quartz near the surface is about $3, but it varies between seventy-five cents and hundreds of dollars per ton.

The quartz contains iron pyrites, copper pyrites and tetrahedrite, sometimes rich in gold, sometimes rich in silver, and then again barren. A depth of about 400 feet has been reached, and, from the continued improvement in the microscopic character of the quartz, and the occurrence of occasional bunches of very rich ore, I am satisfied that a company with money and courage to open to a depth of 1000 feet, as is being done on the mother lode, will meet with success.

The barren block-quartz, while not entirely free from vacuoles, sometimes arranged in parallel bands, usually shows very few. More frequently the quartz is clouded with patches or streaks of iron oxide. In the appearance of its slides, it stands between rich quartz and the chert or jasper that occurs in this district in barren veins.

I have examined slides from Grass Valley, several mines in Amador and Calaveras counties, Cal., in Sonora, Montana, New Mexico, and the newly-discovered quartz veins of Michigan, taking the most prominent mines from which I could get rich ore.

The Idaho, North Star, Empire, Allison, Tiger, Soaproot, Scorpion, Sheep Ranch, Nevill's and Bonanza, represent about all the different characteristic veins of the California gold belt. The photographs (for which I am indebted to Mr. W. H. Walmsley, of Philadelphia) clearly show the arrangement of the vacuoles and markings, more or less parallel in the rich quartz and irregular in the barren. This is so marked in some cases that I hoped that it would prove a good rule to distinguish rich and poor veins. However, the more slides I examined, the less certain it became as a positive sign. Still, there is a chance of error in the apparent exceptions. A poor piece of rich quartz may have been sent to me instead of the quartz from the barren zone. No rich quartz that I have examined is without the parallel arrangement, more or less distinctly defined on some part of the slide, while the greater portion of the barren quartz shows no such markings. The quartz of the enclosing schists does not show parallel vacuoles, but the arrangement of the minerals is parallel.

There seems to be no special difference under polarized light. The rich quartz, however, is usually monochromatic over a larger field, sometimes almost the entire slide, whereas the coloring of the poor slides shows a granular structure.

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From the examination of many rich gold-quartz specimens, the conclusion was formed that the quartz was in a gelatinous condition when the gold was introduced, rather than in the plastic condition of melted slag. A local, circumscribed, intense action must have been the cause of the deposit of rich gold-pockets. In one case, the quartz was covered with a rose-pink coating where the gold was very rich; but, unfortunately, the specimens passed out of my hands before I had a chance to determine its nature. The cavities between the quartz crystals and the gold were filled with dried mud of strong, solfataric smell.

I am not prepared to say that the parallelism is a sure sign of rich ore, but, if I may judge from the material I have collected from the mines visited, the indications are at least worthy of farther investigation. They seem to show that the same causes which crowded the quartz with vacuoles in parallel bands, impregnated the quartz with gold. If it could be proved that all these vacuoles were filled with carbonic acid, it might prove to be a second fact to weave into a theory of the deposition of auriferous quartz.

Many of the slides show peculiar formations along certain bands like streams of little disks of concentric structure. These I take to indicate chalcedonic change of the quartz.

As a rule, the barren quartz has but few vacuoles, seldom arranged in lines. The dark spots shown in the slides, instead of being large vacuoles, are patches of foreign matter.

Slides Nos. 2 and 5 show a peculiar circular polarization between crossed Nicols, due to their chalcedonic nature.

The origin of the material used for the slides, of which photographs are appended, was as follows:

No. 1, Sheep Ranch mine, Calaveras County, Cal. Poor quartz of same kind as rich No. 9.

No. 2, Volcanic quartz from Socorro, New Mexico. It contains no minerals of value.

No. 3, Tiger mine, Calaveras County, Cal. Barren quartz, granular type.

No. 4, Tiger mine, Calaveras County, Cal. Green chert from main shaft, barren cross-seam.

No. 5, Anglo-Saxon mine, Calaveras County, Cal. A barren vein of jasper quartz.

No. 6, Bonanza mine, Sonora, Tuolumne County, Cal. This is said to be from the barren portion of vein, but as it was picked up by the foreman on the dump, there is some doubt. This is very

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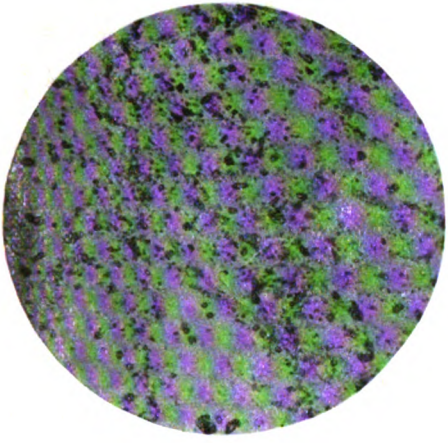
POOR QUARTZ

No 1



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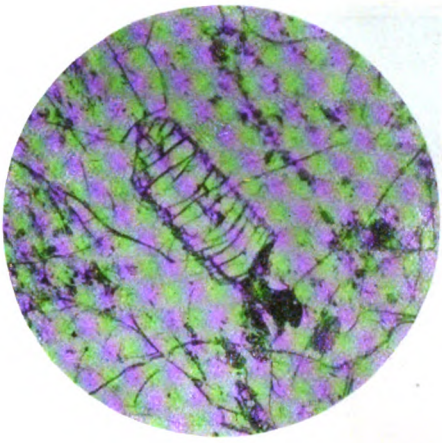

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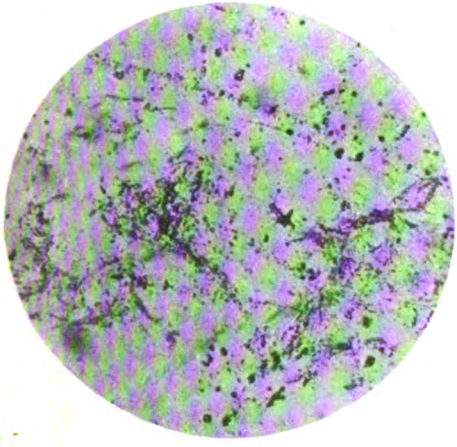

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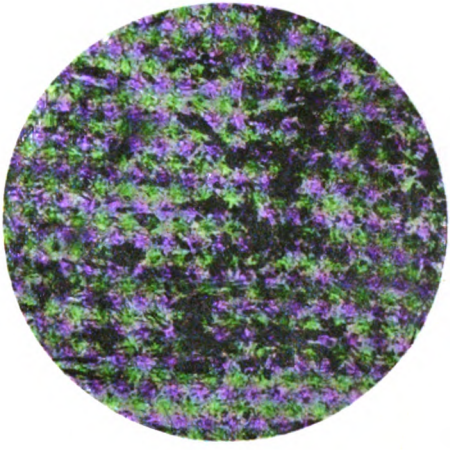

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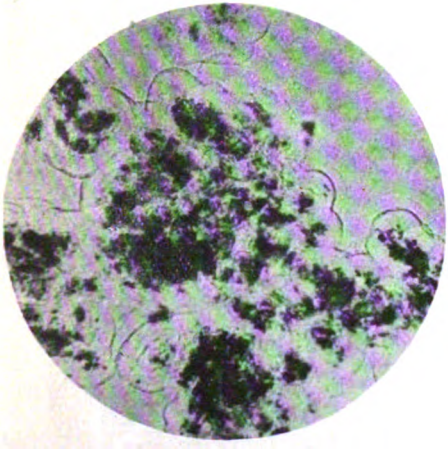

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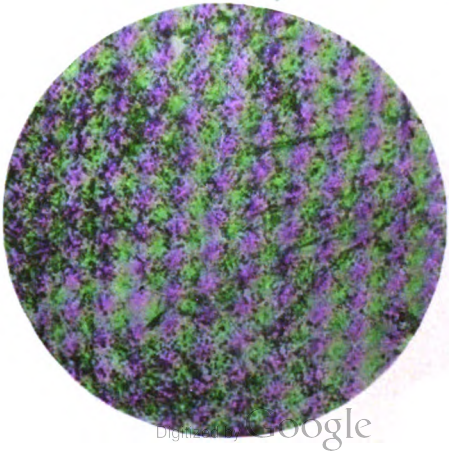

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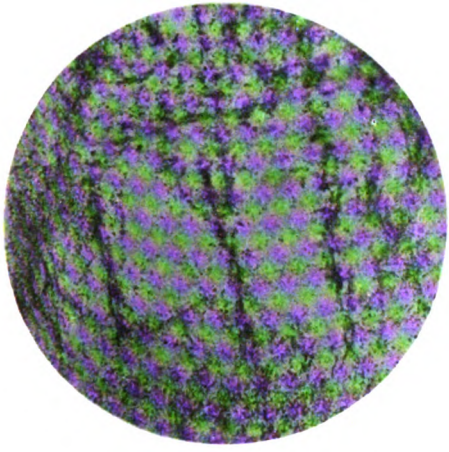
GOOD QUARTZ

No. 7



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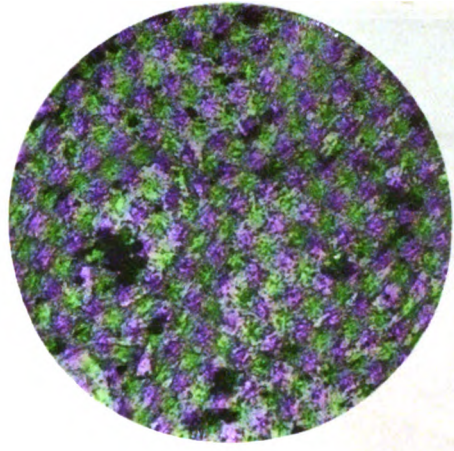

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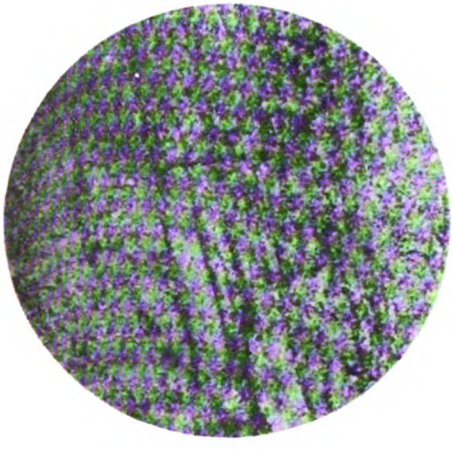

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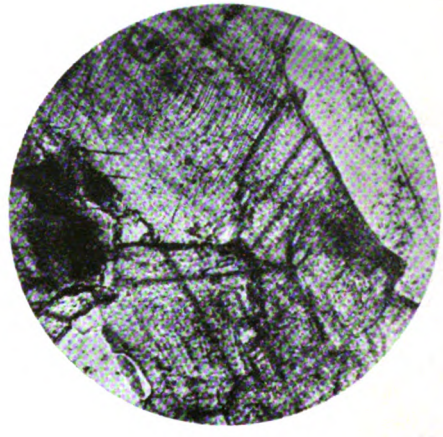

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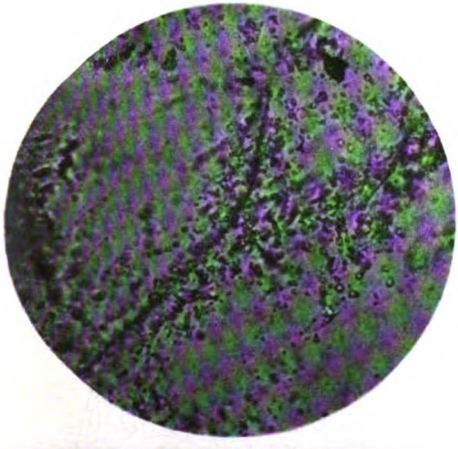

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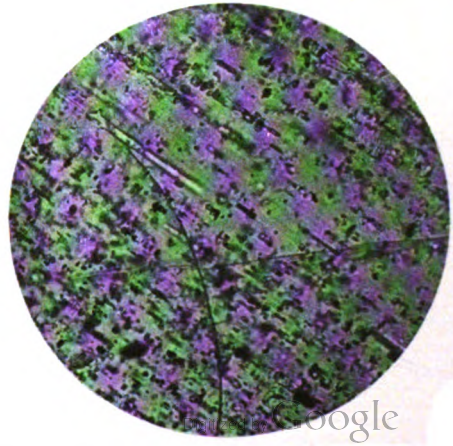

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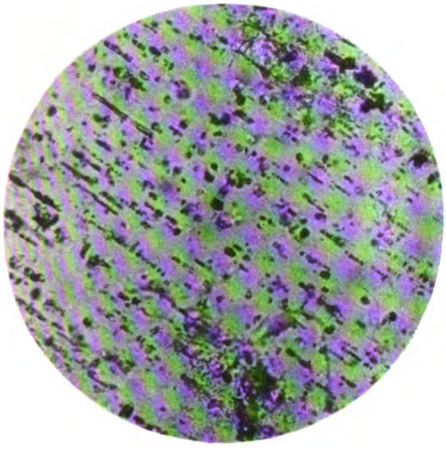
GOOD QUARTZ

No. 13



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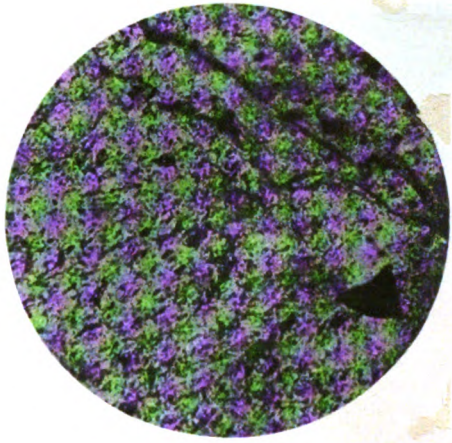

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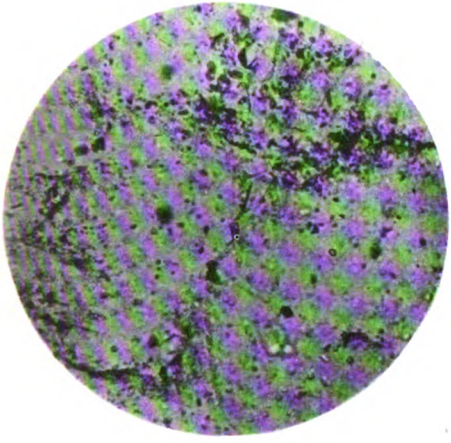

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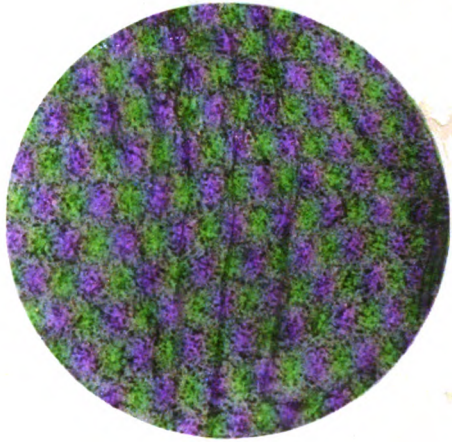

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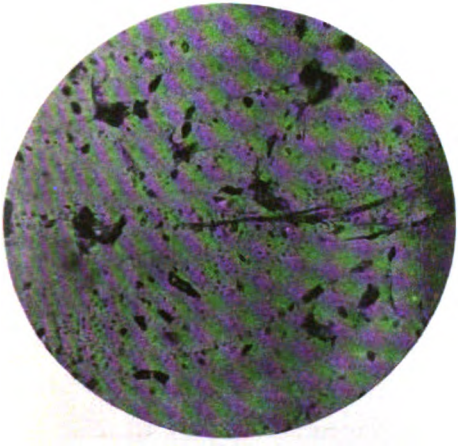
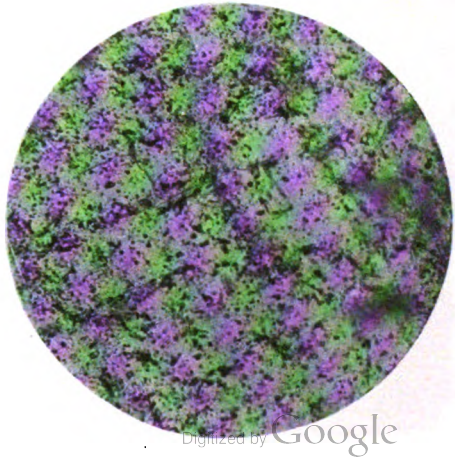


No. 18



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similar to the rich quartz, being parallel in part, although not so evident.

No. 7, Bonanza mine, Sonora, Tuolumne County, Cal. This ore averages $6 to $10 per pound for the month's run, producing $100,000 and upwards each month. Some masses are more than half gold. This is a pocket mine, the other quartz being without value. (No. 6.)

No. 8, Idaho mine, Grass Valley, Nevada County, Cal. This is their usual run of ore, showing free gold. It averages about $22 per ton. Masses of quartz, exceedingly rich, occur.

No. 9, Sheep Ranch mine, Calaveras County, Cal. The sample was the quartz from the specimen ore. All their quartz shows very fine vacuoles, with a general parallel arrangement. No. 1 is poor quartz in the chutes.

No. 10, Nevill's mine, Angels Camp, Calaveras County, Cal. A new strike of good ore, said to be $12 per ton. The mine is on the mother lode, and contains an immense body of low-grade ore, ranging between $3 and $5, not including the ore from the recent new strike.

No. 11, Empire mine, Grass Valley, Nevada County, Cal. Average ore going to stamps.

Nos. 12 and 13, Tiger mine, Calaveras County, Cal. Two parts of the slide. This ore was worth $65 per pound. It occurs in small seams, ½ inch to 3 inches wide. The average of all the quartz is about $3 per ton. The good quartz from all parts of the mine has this character, while the barren block quartz is No. 3.

No. 14, Tiger mine, Calaveras County, Cal. Deep shaft about 400 feet below No. 2. The large dark spot is native gold. Parts of this quartz are nearly as parallel as No. 12. The seam is 6 inches wide, and averages $500 per ton.

No. 15, Gwinn mine, Calaveras County, Cal. Formerly a good mine; shut down for years. Average ore shows no free gold.

No. 16, Soaproot mine, West Point, Calaveras County, Cal. Narrow veins of high-grade sulphurets, with some free gold, in granite walls. The sulphurets run from $75 to $600.

No. 17, Allison mine, Jackson, Amador County, Cal. One of the parallel veins east of the mother lode, in diorite. The ore runs from $3.50 to $12. The specimen shows free gold. The total cost of working this quartz is $1.12 per ton.

No. 18, Scorpion mine, West Point, Calaveras County, Cal. Same in formation as No. 16.

If any of the members of the Institute, in charge of gold mines,

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are sufficiently interested to make slides of the quartz in immediate contact with native gold, and also a slide of the known barren quartz, and send me the result of such examinations, I will embody the results in another paper when I have sufficient data to make a generalization.

THE VELOCITY OF BODIES OF DIFFERENT SPECIFIC GRAVITY FALLING IN WATER.

BY ROBERT H. RICHARDS AND A. E. WOODWARD, BOSTON, MASS.

(Washington Meeting, February, 1890.)

IN Rittinger's *Aufbereitungskunde* of 1867 occurs (p. 195) the following table, which shows the rate of falling in water of fragments of minerals, of irregular shapes and of five different specific gravities:

		Diameter of particles in millimeters.							
	10	8	6	4	3	2	1	.5
	Sp. Gr.	Velocity in meters per second.							
Auriferous silver,	15.	.9	.8	.7	.57	.5	.4	.29	.20
Galena, . . .	7.5	.62	.55	.48	.39	.34	.28	.20	.14
Pyrites, . . .	5.0	.49	.43	.38	.30	.26	.22	.15	.10
Quartz, . . .	2.6	.30	.27	.24	.19	.17	.14	.10	.07
Coal, . . .	1.3	.13	.12	.10	.08	.07	.06	.04	.03

With the purpose of extending this valuable table, to cover a larger number of minerals and sizes of each mineral, and of making it more easy of application in deciding upon a sieve-scale for any group of minerals, we have prepared the curves of velocity of fall of bodies in water shown in Plates I., II., III. and IV., using Rittinger's formula

$$V = 2.44\sqrt{D(\delta - 1)}$$

where V = velocity in millimeters per second.

" D = diameter of particles in meters.

" δ = specific gravity of the mineral.

These curves embrace a large number of specific gravities, and for each specific gravity the relation of the diameter of the mineral particle to the velocity of its fall in water is given for all diameters and velocities used in practice.

The second diagram is an enlargement of the lower left-hand corner of the first, the third bears the same relation to the second, and in like manner the fourth to the third. Plate I, therefore, covers the work of the coarsest jigging 2 inches in diameter; and under

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Plate II, covers from $\frac{1}{2}$ inch downwards; Plate III, from $\frac{1}{10}$ inch downwards; Plate IV, from $\frac{1}{50}$ inch downwards.

We have compiled a table of most of the minerals likely to de-

Substance.	Sp. Gr. δ.	Substance.	Sp. Gr. δ.
Mineral Coal	1.0 – 1 8	Stibnite	4.5
Gypsum	2.3	Tetrahedrite	4.5 5.1
Chrysocolla	2.0 2.2	Bornite	4.4 5.5
Graphite	2.09	Marcasite	4.68 4.85
Quartz	2.5 2.8	Pyrolusite	4.8
Orthoclase	2.44 2.6	Greenockite	4.8 5.0
Serpentine	2.5 2.65	Pyrite	4.8 5.2
Oligoclase	2.56 2.72	Menaccanite	4.5 5.0
Calcite	2.5 2.78	Hematite	4 5 5.3
Labradorite	2.67 2.76	Millerite	4.6 5.65
Talc	2.57 2.8	Magnetite	4.9 5.2
Pyrophyllite	2.75 2.92	Franklinite	5.07
Muscovite	2.75 3.1	Miargyrite	5.2 5.4
Anhydrite	2.9	Proustite	5.4 5.6
Dolomite	2.8 2.9	Chalcocite	5.5 5.8
Margarite	2.99	Zincite	5.4 5.7
Cryolite	2.9 3.07	Cerargyrite	5.55
Amphibole	2.9 3.4	Pyrargyrite	5.7 5.9
Apatite	2.9 3.25	Cuprite	5.85 6.15
Tourmaline	2.9 3.3	Scheelite	5.9 6.1
Fluorite	3.0 3.25	Cobaltite	6 0 6.3
Forsterite	3.3 3.5	Arsenopyrite	6.0 6.4
Pyroxene	3.2 3.5	Tellurium	6.1 6.3
Epidote	3.25 3.5	Anglesite	6.1 6.4
Fowlerite	3.4	Wulfenite	6.0 7.0
Rhodonite	3.4 3.68	Polybasite	6.2
Rhodochrosite	3.4 3.7	Stephenite	6.27
Azurite	3.5 3.8	Cassiterite	6.4 7.1
Diamond	3.53	Smaltite	6.4 7.2
Calamine	3.16 3.9	Cerussite	6.5
Garnet	3.15 4.3	Pyromorphite	6.5 7.1
Siderite	3.7 3.9	Wolframite	7.1 7.55
Limonite	3.6 4.0	Argentite	7.19 7.36
Malachite	3.7 4.0	Galenite	7.25 7.7
Atacamite	3.8	Niccolite	7.3 7.7
Sphalerite	3.9 4.2	Sylvanite	7.99 8.3
Corundum	3.9 4.2	Copper	8.84
Willemite	3.9 4.18	Cinnabar	9.0
Tephroite	4.0 4.1	Silver	10.1 11.1
Smithsonite	4.0 4.5	Amalgam	10.5 14.0
Chalcopyrite	4.1 4.3	Palladium	11.3 11.8
Chromite	4.3 4.57	Mercury	13.57
Barite	4.3 4.7	Gold	15.6 19.5
Enargite	4.4 4.45	Platinum	16.0 19.0
Pyrrhotite	4.4 4.7	Iridosmine	19.3 21.12
Molybdenite	4.4 4.8	Platiniridium	22.6 23.00

mand water-separation, using the specific gravities given in Dana's *System of Mineralogy*.

The diagrams may be used in determining, for any group of

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minerals to be separated, a sieve-scale, that is to say, a list of sieves arranged in series in such manner that the sands treated by them shall be suitably prepared for jigging. Rittinger (p. 224) adopts as the most satisfactory sieve-scale a list of sieves beginning with one having 1 mm. hole and ranging upwards in geometric progression by the factor $1.4142 = \sqrt{2}$ and downwards by the reciprocal $\frac{1}{1.4142}$. He proceeds to show (p. 271) that not all the sieves of this scale are needed for separating two minerals of greatly diverse specific gravity, for instance, quartz and galena. This may be deduced from the following table, constructed from his figures:

	Rittinger's Sieve-Scale (p. 224).	Rittinger's Fall-velocities (p. 271). Velocity of Fall in Meters per Second.	
	Diameter in Millimeters.	Galena, sp. gr., 7.5.	Quartz, sp. gr., 2.6.
Group I.	64.0	1.571	0.780
	45.2	1.322	.671
	32.0	1.112	.551
Group II.	22.6	.922	.457
	16.0	.786	.390
	11.3	.661	.328
	8.0	.556	.276
Group III.	5.6	.463	.230
	4.0	.392	.195
	2.8	.327	.161
	2.0	.278	.137
Group IV.	1.4	.232	.115
	1.0	.195	.097
	0.71	.165	.082
	0.50	.138	.073
Group V.	0.35	.116	.058
	0.25	.098	.048
	0.125	.069	.034

Group VI. comprises fine slimes.

The smallest grain of galena in any one group will settle more rapidly than the largest grain of quartz in that group.

The sieve-scale required for quartz and galena, as shown by the above table, is then:

Group.	Mesh in millimeters.	
I.	through 64.	on 22.6
II.	" 22.6	" 5.6
III.	" 5.6	" 1.4
IV.	" 1.4	" 0.35
V.	" 0.35	" 0.125
VI.	" 0.125	

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It is to facilitate the making of such shortened sieve-scales, adapted to special groups of minerals, that the fall-curve diagrams have been constructed. For instance, if it be desired to construct a sieve-scale for the preparation of sands containing quartz (sp. gr., 2.6) and apatite (sp. gr., 3) for jigging, we start with the proviso that in any given group the smallest grain of apatite must have one centimeter per second greater velocity than the largest grain of quartz in that group. Taking one centimeter = 0.01 meter diameter as the limiting sieve under the rolls, we can then select by inspection from the diagram (Table 2), the following sieve-scale:

Class.	Mesh in Meters.		Velocity in Meters per Second.	
	Through.	Upon.	Largest Quartz.	Smallest Apatite.
I.	0.01	0.0084	0.304	0.316
II.	0.0084	0.0070	0.280	0.290
III.	0.0070	0.0060	0.256	0.266
IV.	0.0060	0.0050	0.236	0.245
V.	0.0050	0.0042	0.215	0.225

And so on. A similar sieve-scale for classifying, preparatory to jigging, the grains of any two minerals of different specific gravities, can be made from the diagrams, with less trouble, we think, than in any other way.

The velocities given in the curves are all of them final velocities calculated by Rittinger's formula. In quick jigging the conditions of his formula would not be quite realized as the particles would not have time to acquire their final velocity. To give some idea to what extent the curves would be in error for quick jigging we add the following table taken from Rittinger, page 178, but much enlarged.

$$V = \frac{1}{A} \cdot \frac{e^{2Bt}-1}{e^{2Bt}+1} \text{ when } B = \frac{g(s-\triangle)}{s} A \text{ and } A = \sqrt{\frac{3\,\alpha_3\,\triangle}{2dj\,(s-\triangle)}}.$$

V = velocity of fall in meters per second.
e = Napierian base.
g = acceleration due to gravity.
s = sp. gr. of solid.
$\triangle$ = sp. gr. of liquid.

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d = diameter of solid.
j = weight of cu. in. of water.
a_3 = constant velocity v after t in seconds.

Substance.	Diameter in meters.	Fall in meters per second.					
		$t = 0.062$.	$t = 0.125$.	$t = 0.25$.	$t = 0.50$.	$t = 1.0$.	$t = 2.0$.
PbS	.016 ($\frac{5}{8}$ in)	0.514	0.936	*1.411*	*1.630*	*1.650*	*1.650*
FeS_2		0.468	*0.825*	*1.174*	*1.287*	*1.293*	*1.293*
SiO_2		0 353	*0.570*	*0.767*	*0.801*	*0.817*	*0.817*
PbS	.012 ($\frac{1}{2}$ in.)	0.508	0.902	1.290	1.388	1.428	1.428
FeS_2		0.461	0.789	1.054	1.118	1.120	1.120
SiO_2		0.345	0.558	0.689	0.708	0.709	0.709
PbS	.008 ($\frac{3}{8}$ in.)	0.497	0.842	1.107	1.164	1.166	1.166
FeS_2		0.448	0.723	0.890	0.914	0.915	0.915
SiO_2		0.332	0.499	0.572	0.578	0.579	0.579
PbS	.006 ($\frac{1}{4}$ in.)	0.487	0.790	0.980	1.009	1.010	1.010
FeS_2		0.436	0 669	0.778	0.792	0.792	0.792
SiO_2		0.319	0.454	0.498	0.501	0.501	0.501
PbS	.004 ($\frac{1}{6}$ in.)	0.468	*0.704*	*0.814*	*0.823*	*0.824*	*0.824*
FeS_2		0.414	*0.586*	*0.643*	*0.646*	*0.646*	*0.646*
SiO_2		0.297	*0.383*	*0.409*	*0.409*	*0.409*	*0.409*
PbS	.003 ($\frac{1}{8}$ in.)	0.451	0.645	0.710	0.714	0.714	0.714
FeS_2		0.394	0.527	0.559	0.560	0.560	0.560
SiO_2		0.282	0.344	0.354	0.354	0.354	0.354
PbS	.002 ($\frac{1}{12}$ in.)	0.421	0.553	0.582	0.583	0.583	0.583
FeS_2		0.362	0.445	0.457	0.457	0.457	0.457
SiO_2		0.250	0.286	0.289	0.289	0.289	0.289
PbS	.001 ($\frac{1}{25}$ in.)	0.354	*0.409*	*0.413*	*0.414*	*0.414*	*0.414*
FeS_2		0.294	*0.321*	*0.323*	*0.323*	*0.323*	*0.323*
SiO_2		0.195	*0.203*	*0.204*	*0.204*	*0.204*	*0.204*
PbS	.0005 ($\frac{1}{50}$ in.)	0.277	0.291	0.292	0.292	0.292	0.292
FeS_2		0.222	0.229	0.229	0.229	0.229	0.229
SiO_2		0.143	0.145	0.145	0.145	0.145	0.145

The heavy-face italic figures were taken from Rittinger, the others were calculated.

All jigging will practically take place between $t = 0.25$, or 240 strokes per minute, and $t = 0.50$, or 120 strokes per minute. Now if we follow down these two columns in the above table we shall see that this error is serious for 240 strokes, with $\frac{3}{8}$-inch, $\frac{1}{2}$-inch and $\frac{1}{3}$-inch diameter; becomes insignificant with $\frac{1}{4}$-inch, $\frac{1}{6}$-inch and $\frac{1}{8}$-inch, and disappears practically in $\frac{1}{12}$-inch, $\frac{1}{25}$-inch and $\frac{1}{50}$-inch diameter; that for 120 strokes per minute it is nowhere serious; is insignificant for $\frac{3}{8}$-inch, $\frac{1}{2}$-inch and $\frac{1}{3}$-inch diameter, and practically disappears for $\frac{1}{4}$-inch and all below.

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LIST OF COMMERCIAL PHOSPHATES.

BY W. H. ADAMS, NEW YORK CITY.

(Ottawa Meeting, October, 1889.)

INASMUCH as the mining of phosphate rock and its transportation has so general a bearing on the great industries of this continent, entering as well into the problem of our trade with foreign countries, it may be useful to place on record the following list of localities from which certain products of a like nature are received by us in a business way, or in which we are interested more or less either by ownership or by competition.

As a ready means of comparison, the percentages of the mine products and the deteriorating constituents are given:

Coprolite (Cambridge).—This material is obtained from the Upper Greensand formation in Cambridgeshire. The phosphate runs from 50 to 60 per cent., and averages 54 per cent. It contains much carbonate of calcium, but little iron, in the ferric condition. The superphosphates manufactured from the material are permanent and little liable to "revert." They contain from 28 to 35 per cent. of soluble phosphate.

Coprolite (Buckingham).—From the same formation as the Cambridge variety, and very similar in composition. It averages 60 per cent. phosphate.

Coprolite (Suffolk).—These coprolites are not so valuable for the manufacture of superphosphate as the two previous varieties, as the iron they contain is in the ferric state, and liable to "revert" the soluble phosphate. The phosphate varies from 48 to 60 per cent., but averages 54 per cent. They are of Tertiary formation.

Coprolite (Potton or Bedford).—The average amount of phosphate in these coprolites is about 50 per cent. In some cases they may, however, run up to 60 per cent., and in others may not exceed 45 per cent. They belong to the Lower Greensand formation. The ferric iron frequently is as high as 4 per cent., and the alumina as much.

Russian phosphate comes from the chalk, and occurs in curious rounded masses. It contains from 28 to 35 per cent. of phosphate, but may run up to 69 or 70 per cent.

Curaçoa phosphate is also of very high class quality, giving 80

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to 88 per cent. of phosphates. It dissolves well to a first-class compound of from 40 to 45 per cent. soluble phosphate.

Fossil guano phosphate, obtained in a fine state of division, varies much in its composition. Of two analyses, one contains only 22 per cent. of phosphate, while the other holds 36 per cent.

Cuban phosphate contains from 45 to 50 per cent. of phosphate, accompanied by from $1\frac{1}{2}$ to $2\frac{1}{2}$ per cent. of iron and alumina. It also gives about 1 per cent. of ammonia.

Redonda phosphate, from the Leeward Islands, is a hydrated phosphate of alumina, containing about 44 per cent. of phosphoric acid (which is equal to about 70 per cent. of calcium phosphate), about 6 per cent. of ferric iron, and 20 per cent. of alumina. It is the subject of several patents, which had for their object the conversion of the body into sulphate of alumina and phosphate of calcium.

Boulogne phosphate is a coprolite containing much silica, and from 40 to 45 per cent. of phosphate. It yields permanent dissolved compounds.

Sombrero Phosphate.—This is one of the best of the many phosphates yielded us by the Windward Islands of the Caribbean Sea. It is practically free from iron, and yields the highest class of superphosphates. The phosphate contained is equal to from 65 to 78 per cent., and averages high.

Navassa phosphate or Cooperite consists of a mixture of phosphate of calcium and magnesium, of which it contains from 47 to 56 per cent. It also contains from 4 to 5 per cent. of ferric iron, and from 8 to 10 per cent. of alumina, and is therefore of low class for dissolving.

Antwerp or Belgian phosphate is shipped in a fine state of division. It is a low class phosphate of from 30 to 35 per cent.

Aruba phosphate, from the Windward Islands, is a phosphate very free from iron, and dissolves well. It is of very high class quality, yielding from 75 to 85 per cent. of phosphate, and giving a superphosphate of from 35 to 37 per cent. soluble phosphate.

Phosphorite or Estramadurite.—This is a Spanish phosphate, containing 76 to 82 per cent. of true phosphate. Occasionally it falls below 50 per cent., and in such cases will contain either limestone or quartz.

Welsh phosphorite is found in the Silurian system (Lower). It gives 50 to 60 per cent. phosphate; it is dark in color, and often contains much carbonate of calcium.

Nassau or German phosphate is from the Devonian system of

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Nassau. It varies much in composition, running in phosphates from 30 to 75 per cent. It contains a considerable proportion of iron—up to 6 per cent. of ferric oxide—and alumina—often 4 per cent.—and is therefore not in great repute on account of the large amount of "reverted" phosphate formed in the superphosphate.

Norwegian apatite frequently reaches 90 per cent. of phosphate of calcium. It is one of the highest class phosphates.

In the same connection it might be well to call attention to the rapid strides made in the introduction of the phosphatic slags, from the Thomas-Gilchrist process, in Europe and in America.

Such inroads have been made in the demands for our own mine-phosphates since this slag has been introduced and endorsed by many of the most eminent agricultural chemists of the old world as to bring about very close study of the constituents of all the slags which this process has given us from different phosphoritic iron-ores.

At the Birmingham meeting, May, 1888, an article on "Phosphate Slag" * was read by William B. Phillips, Ph.D., which contains an excellent digest of the subject, and is a valuable contribution, with conclusions which raised the hopes of producers, and have no doubt given a certain impetus to the many schemes for introduction of the phosphoritic ores of the southern States.

The experiments, however, on both sides of the Atlantic, with all the stimulus of competition, have so far developed the fact that the best of slags have not over one-half the value of acid phosphate. Therefore there is less danger of the introduction of slags in a large way than of the low-grade marl-phosphates which are being discovered along the James River, in Virginia, or in several sections of North Carolina, Florida, etc.

It is estimated that Germany will dispose of 250,000 tons, and England 150,000 tons of slags this year, and the statement is made that Prussia sold 150,000 tons the past year, at an average price of $8.00 per gross ton. It is claimed that in the latter case, with the addition of 5 pounds of saltpeter to each 100 pounds of the slag (Schlacke), the mixture is fully equal to the best bone-manure, and that a value of $2.25 of this material will put an acre of land in first-class condition, after which a less quantity is required.

As opposed to this statement, certain chemists declare that the amount of phosphoric acid in average slags sold of late is not over 12 to 14 per cent., with say 3 to 4 per cent. soluble, and that prices

* *Trans.*, xvii., 84.

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charged as above are therefore altogether too high, as compared to phosphate in mine-rock.

This struggle between the chemists of the rival manufacturing concerns is productive of much good to the general public, as it brings prominently before us the values of our present sale-products, and introduces possible supplies which we are to depend upon in future. It is also very gratifying to note that the controversies and friction between the manufacturers over processes, values, etc., has gradually brought about a change in the control of the business from rule-of-thumb operators to that of educated chemists. It can only be stated at this time that "a slag is not a slag," the statements of the selling agents notwithstanding, but it is hoped that soon such correctly prepared tables will be presented as shall give exact values of all low-grade phosphatic materials which are before the public, including slags from various sections of our own country.

M. Edwin Jensch, in the *Zeitschrift für Angewandte Chemie*, June 1, 1889, shows that one of the factors of fertilization which slags contain, but which has received little attention heretofore, is the basic silicate of lime, which is very soluble in organic acids, this being a property which is completely absent from the siliceous constituents of mineral phosphates; also that the action of weak acids on Thomas slags (tetrabasic phosphate) is related to the greater or less proportion of combined silica. The more acid the silicate, the less soluble, and this is true also of the phosphoric acid.

Another constituent of slag which is of great importance in determining its value, *i.e.*, the magnesia, should always be taken into account if the proper idea of slag-values be obtained.

Magnesium, like calcium, forms a superphosphate, but its solubility is greatly in advance of that of the lime salt, therefore more rapidly dissolved by the action of organic acids in the soil. Jensch clearly shows that the soluble acid in rich slags is sometimes one-third, always one-quarter, of the total acid, whereas in the poor slags it averages no more than one-sixth.

A step in advance has already been hinted at by English engineers, viz.: the absolute and controllable union of phosphorus with an aluminous base *in the furnace*—a process by itself and carried on so understandingly that either for sale direct, or for subsequent treatment to develop new combinations, the slags shall have a definite value.

The proposition is to pulverize rich phosphatic slags, then mix with salt and coke-dust and form briquettes. These briquettes are

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placed in a furnace and covered lightly with screened coke, the mass raised to a red heat and a blast of steam introduced.

The re-actions produce phosphate and aluminate of soda, which are separated by lixiviation and crystallization.

GEOLOGY OF THE CHOCTAW COAL-FIELD.

BY H. M. CHANCE, PHILADELPHIA, PA.

(Washington Meeting, February, 1890.)

THE Choctaw coal-field is a direct westward extension of the Arkansas coal-field, but its coals are not like Arkansas coals, except in the country immediately adjoining the Arkansas line.

From the base of the coal-bearing rocks up to the top of the coal-measures, I find a total thickness of at least 8500 feet. This great mass of coal-bearing rocks consists of an alternation of slates, shales, sandstones and coal-beds, with their accompanying under-beds of fire-clay. Only one small bed of limestone was observed. This occurs near the middle of the series; it is about eighteen inches thick, and quite arenaceous.

The formation is naturally subdivided by seven or eight thick beds of sandstone, varying from 50 to 200 feet in thickness, the outcropping edges of which form a series of more or less bold "hog-back" ridges, the interbedded shales and slates forming the intervening valleys.

The base of the coal-series is a massive sandstone, ranging from 100 to 200 feet or more in thickness, lying immediately beneath the Grady coal-bed, which is the lowest known coal. In the district embraced between the Missouri, Kansas and Texas railroad and the Arkansas State line, this sand-rock usually forms a bold semi-mountainous ridge.

This is the ridge through which the St. Louis and San Francisco railroad passes at Bryan station, where the Grady coal-bed is opened and worked, from which point it can be traced westward without difficulty, passing about three miles north of Le Flor on the same railroad, thence west to the "Little Narrows" (which is merely a gap in the ridge), and beyond to a point two miles west of the Thompson-McKinney place, where it swings abruptly north for a mile or more, only to resume immediately its westward course, forming for some miles the northern boundary of the valley known as the "Boiling-Springs Prairie," beyond which it trends somewhat

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southwest, crossing Gaines's creek, thence west to, and south of, the Simpson place (two or three miles east of the mining town of Krebs), where it abruptly turns back, running in an easterly course about 7 miles to Brushy creek, which it crosses, maintaining its eastward course for a further distance of about 6 miles, here forming the southern boundary of the Boiling-Springs prairie, and the northern boundary of the Grady coal-basin. From the point last reached, the ridge swings around in a curve to the southwest, a distance of about 9 or 10 miles, inclosing and forming the valley and coal-basin which I have named, after its discoverer, the Grady coal-basin.

In tracing this ridge eastwardly from the Bryan mine, we follow it without difficulty to and beyond the Poteau river, to a point about 5 miles nearly due south from Cavaniol station. Here it turns abruptly back to the southwest, and, swinging in a horse-shoe curve to the south and southeast, incloses a basin which I have called the Mitchell basin, naming it likewise after its discoverer.

The discovery of this basin furnishes a good example of the results that can, at times, be reached by the structural geologist in predicting, in advance of the actual examination of a district, what may possibly be found. Having traced this ridge to the Poteau river, and from an elevated point being able to see that it was running straight towards the Poteau mountains, some 10 or 15 miles to the east, and knowing that these mountains were formed of rocks high in the coal-measures, I saw at once that either a great fault must cut off these lower rocks, or that they must turn back to the west or southwest, and then, resuming their easterly course, pass south of the Poteau mountains. The first hypothesis being deemed improbable, the second was assumed to be probably true, in which case, a coal-basin should be found a few miles south of our camp on the Poteau river. Just as I was about starting in search of this basin, a courier reached camp with a message requiring my immediate return to McAlester. Upon leaving camp, I gave my chief prospector (Mitchell) directions how to proceed in search of this basin, which he afterwards found, without trouble, located exactly as I had predicted.

A survey-line, started from the ridge at Bryan, and running nearly north, directly across the dip, to the Kavanaugh mountains, north of Kennedy's store, a distance of about 7 miles, would give the data for a cross-section, from which the thickness of the coal-measures could be obtained with accuracy. At Bryan the dip is

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about 40°; at 1 mile north, about 30°; at 2 miles north, about 20°; and for the next 2 miles, about 15° to 20°. Still farther north it gradually decreases, until under the main ridge or "backbone" of the Kavanaugh mountains, the rocks are flat. A section, roughly constructed from odometer measurements, gives a minimum thickness of 8500 feet, with a possibility of 10,000 feet, of coal-bearing rocks in this basin.

Should the entire series prove of Carboniferous age the thickness here present is about twice as great as has yet been found in any State in the Union. Probably about the same thickness will be found in Arkansas, but up to the time my Indian Territory work was finished Mr. Winslow's work for the Arkansas Geological Survey had not progressed to a position warranting any definite statements as to total thickness, hence we were not able to compare results, but he already evidently anticipated the existence of a thickness far in excess of that shown in other States of the Union, and was not surprised to hear of a similar occurrence in the Indian Territory.

The coals worked in the McAlester, Savannah and Lehigh districts, on the Missouri, Kansas and Texas railroad, and at Bryan, on the St. Louis and San Francisco railroad, belong to the lower portion of this 8500 feet, being principally beds found within 1500 feet of the basal sand-rock.

The coals of the Kavanaugh mountains, notably the big bed at Mayberry's mine, and those in the mountain near Poteau switch, on the St. Louis and San Francisco railroad, belong high up in the series, being found in the uppermost 1500 feet of rock. To this series, also, probably belong the coals opened and mined in the western part of Arkansas at Jenny Lind, Hackett City and Huntington.

The western Arkansas coals are dry semi-bituminous or semi-anthracitic coals, mostly non-coking, or with quite feeble coking-properties, ranging from 14 to 16 per cent. in volatile matter, the highest percentage yet found, according to Mr. Winslow's Arkansas report, being 17.655.

In the Mitchell basin, about 10 miles west from the Arkansas line, coal recently opened shows 19 per cent. volatile matter; the Mayberry coal, about 8 miles farther west, contains 23 per cent. volatile matter, and the Bryan mine coal, about the same distance west, shows 26 per cent. volatile matter. About 30 miles farther west, the coal shows from 38 to 41½ per cent. volatile matter, which

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is also about the percentage in coals of the McAlester and Lehigh districts.

The Mitchell basin coal will coke; but whether it will make marketable coke has not yet been determined. From the Bryan mine west, all the coals are strong coking coals; but they contain so much gas that the yield of coke will be small, except in the district between the Bryan mine and the Little Narrows.

The coals best adapted for general use, are found in the district lying at and west of the Little Narrows, in the Grady basin and in the McAlester field. The Mitchell basin coal, now being prospected, also promises coal of good quality, but of semi-bituminous character. A recent analysis, made by McCreath, gave:

	Per cent.
Water,	1.058
Volatile matter,	19.032
Fixed carbon,	71.736
Sulphur,	.649
Ash,	7.525
Total,	100.000

The bed from which this sample was taken, is supposed to be either the Grady or the McAlester bed. The opening was made since my return, and I cannot locate it.

An average of seven analyses, made by Mr. McCreath, of coal from the Grady bed in the Grady basin, shows:

	Per cent.
Water,	1.792
Volatile matter,	40.207
Fixed carbon,	51.785
Sulphur,	1.333
Ash,	4.883
Total,	100.000

Two analyses of coal from the Grady bed in the McKinney (Little Narrows) district, average:

	Per cent.
Water,	1.709
Volatile matter,	38.668
Fixed carbon,	51.482
Sulphur,	1.006
Ash,	7.135
Total,	100.000

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One analysis of coal from the McAlester bed at Krebs (near McAlester) gave:

	Per cent.
Water,	1.804
Volatile Matter,	37.171
Fixed Carbon,	53.404
Sulphur,	.896
Ash,	6.725
Total,	100.000

One analysis from the Lehigh mines gave:

	Per cent.
Water,	4.323
Volatile Matter,	40.507
Fixed Carbon,	44.472
Sulphur,	2.598
Ash,	8.100
Total,	100.000

Both the Lehigh and Bryan mines produce coal objectionably high in sulphur; and the Lehigh coal has, moreover, the high percentage of moisture so common in the coals of Kansas and Missouri. The best coal now mined in the region is that from the McAlester bed, mined at McAlester and Krebs by the Osage Mining Company, and at Alderson by the Choctaw Coal and Railway Company, and that from the Grady bed just opened at Hartshorne by the latter company. These coals compare favorably with the best gas-coals mined in the country (as comparison with standard Pittsburgh coal will show) and they are by far the best coals now mined in the southwest, if not indeed the best mined west of the Mississippi river. They are in every way vastly superior to Kansas, Missouri and Iowa coals.

Topographically and structurally the Choctaw coal-fields represent in miniature many of the features of the anthracite regions of Pennsylvania. The measures are flexed by a series of anticlinal and synclinal folds, not usually as sharp as those of the anthracite regions, but in many respects very similar. While all the anthracite basins are surrounded by a mountainous rim, of which the outcrop of the thick and massive conglomerate forms the core, the Choctaw basins are enclosed by a ridge, sharp and bold in places, but rarely mountainous, formed by the outcrop of the basal sandstone. Whether this rock is the equivalent of the conglomerate or not is as yet wholly conjectural.

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I have been enabled to compile the following generalized section of the coal-measures from measurements rudely made and at points widely separated. In a general way it will be of service to the prospector or field-geologist working in other portions of this coal-field.

COLUMNAR SECTION OF THE COAL MEASURES.

	Feet.
Shales and slates with two massive sandstone ledges. This group forms the back-bone of the Kavanaugh mountains north of Kennedy's store. Whether it contains coal-beds is not now known. Its thickness is estimated at . . .	1200
Mayberry Coal, ...	4 to 6
(This appears to be the Huntington Jenny Lind Coal.)	
Slates, sandstones and shales, with some coal-beds; none, however, known of workable size and quality; this group contains four or five massive sandstones. Its thickness is estimated at about	3500
Slates with two massive sandstones (these form the top rocks in the centre of the basin 4 miles southwest from Frink's Switch on the M. K. & T. R. R.), ...	400
Sandstone, ...	50
Slate, ...	100
Secor Coal, ...	2½
Slate, ...	30
Sandstone, ...	50
Slates, ...	320
Sandstone (forms ridge south of South McAlester), . . .	200
Slates with coal-bed thin, ...	220
Sandstone (ridge at South McAlester), ...	100
Slates with sandstones, ...	200
Sandstones (ridge between McAlester and South McAlester), .	50
Slates, ...	130
Norman Coal, ...	3
Slates with sandstone bed near top and two or three thin coals,	600
McAlester Coal, ...	4
Sandstone, ...	50
Slates and shales with thin coal-bed, ...	500
Sandstone, massive, caps Round-top and Long mountain in the Grady basin, forms bold ridge ½ mile north of Alderson Station on C. C. & Ry. Co.'s railroad "Flat-top Sandstone,"	100
Slates and shales with two thin coals in upper part, . .	600
Upper Grady coal, 1 to 3 ft. Slates or sandstone, 50 ft. Middle Grady coal, 1 to 3 ft. } Grady Group of Coals, . . Slates or sandstone, 50 ft. Grady coal, 4 to 6 ft.	100
Sandstone, massive "Tobucksy Sandstone,"	200

Below this bottom sandrock we find shales and thin-bedded sand-

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stones forming the Fourche valley, and including the "Limestone ridge," which is finely seen at Limestone gap on the M. K. & T. R.R., and on the south side of the valley opposite the Little Narrows. These measures and the included limestone are probably of Subcarboniferous age.

The maps illustrating this paper were made from buck-board odometer measurements, checked by the alignment survey of the Choctaw Coal and Railway Company. The contoured map was made from pocket-compass surveys and barometric levels; but both were thoroughly checked and all large errors were detected and eliminated. The Grady coal-basin, shown on the small map, is a beautiful little basin, enclosed on three sides by the ridge formed by the basal sand-rock (Tobucksy sandstone), and on the fourth by the Adams ridge (formed by the "Flat-top sandstone"), with but four narrow gaps through which entrance or egress may be had. A large part of the basin is flat or gently rolling, partly prairie and partly wooded. Near the center of this basin (exactly in its geological center) are three "mountains" rising abruptly to a height of about two hundred feet, almost perfectly flat on top and capped by a hard, massive sandstone, 30 feet or more in thickness, which outcrops in bold cliffs on all sides, making access to the top both difficult and dangerous.

The maximum depth of the Grady coal-bed in this basin is about 600 feet; but over three-fourths of the basin the bed can be reached at depths less than 450 feet, and over one-half of the basin the depth will probably not exceed 300 feet.

The basin is about 6 miles long by 3 or 4 wide, and contains over 11,000 acres of the Grady bed. Throughout this area the coal is not always of workable thickness; but over a large portion of it the bed will range from 3½ to 5 feet thick, yielding an average of 4 feet of clear coal.

The large map shows the outcrop of the McAlester bed, as now worked between McAlester and Krebs, and its continuation to Alderson and beyond to Brushy creek, where it swings around to the southwest and is lost in the badly-faulted and contorted measures that occupy the country drained by Brushy creek above its junction with Limestone creek.

The Mitchell basin is not accurately located. Just where the Grady and McAlester beds should cross the line into Arkansas is not yet known; and, until this connection is made, the course of the outcrops of these two beds in the country immediately adjacent cannot accurately be defined.

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The axial line of the Kavanaugh mountains is a synclinal with a gentle anticlinal roll occupying the valley of Brazil creek, and another similar roll coinciding very nearly with the course of the Poteau river from Cavaniol to and beyond Poteau station. These anticlinal rolls apparently have a general course of S. 40° W., while the main synclinals run about S. 70° W. to S. 80° W. The same feature was noticed in the McAlester and Grady basin districts, and seems to indicate two distinct types of disturbances, or, what is perhaps more probable, a single flexing force, giving rise to two series of rolls, along the lines of two distinct resultants.

Whatever the cause, the existence of two series of anticlinal and synclinal folds, not parallel, but forming an angle of 30° or 40°, is a fact that should not be overlooked, for it has a most important bearing upon the shape of the basins, and upon the location of those minor disturbances—rolls and faulted areas—an intelligent anticipation of which may often avert the loss of large sums expended in attempting to open and operate collieries located in or near such a disturbed area.

In a general way, we may expect faulted coal, rolls and swamps, at and near the intersection of any two synclinal or anticlinal axes. Thus, such an area is found along Brushy creek, immediately W.S.W. from the Grady basin. At the southwest corner of the Grady basin the "butt-end" of an anticlinal axis is plainly shown by the indentation of the southern rim immediately west of Hartshorne station. This axis apparently runs about S. 20° or 25° W.

In prospecting for coal in this field, the first object is, of course, to find a bed of good quality, and thick enough for profitable mining. As in the McAlester and Lehigh districts coal averaging almost 4 feet in thickness is mined, and as from 4 to 6 feet of coal is worked in the Arkansas fields, a thickness of not less than 3 feet and 6 inches is needed to place a new operation on anything like an equal footing with these older companies.

In the area covered by the map, I have found only three beds reaching or exceeding this thickness, viz.: the Grady, the McAlester and the Mayberry beds. Other beds, such as the Norman or Secor beds, may locally attain good workable thickness; but I do not know that any such areas exist. Hence—barring exceptional occurrences of this kind—the search for workable coal should be confined to the outcrop-lines of these three beds, as shown by the map.

Cheap mining, and the production of as large a percentage of lump-coal as possible, require, also, that the dip of the coal shall be mod-

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erate, not exceeding a pitch of about 14° or 15°, so that the mine-cars may be taken directly to the face of the breasts or rooms. A pitch of 6° or 8° is considered most advantageous, as on such a pitch slopes work well, and the cars are readily taken up to the working-face. When the dip is less than 6° or 8°, the best method of development is by shafts.

Hence, in searching for coal in this field, the prospector rapidly passes, as undesirable, all territory in which the rocks show a strong angle of dip, and, upon reaching an area of moderate dip, his search may profitably be confined to the outcrops of the three beds above named.

The Boiling Springs prairie, and the same valley farther east, where it is known as the Fourche Melane valley, or valley of the "Big Fourche," is not coal-territory. It is occupied by the shales associated with the limestone of "Limestone Ridge," which are possibly of Subcarboniferous age.

SPECIFICATIONS FOR CAST-IRON COATED WATER-PIPE.

BY THOMAS W. YARDLEY, CHICAGO, ILL.

(Washington Meeting, February, 1890.)

In many years' experience as a maker and purchaser of cast-iron coated pipe, I have never met with any standard form of specifications for such. Each water-works company, employing a hydraulic engineer, has, under his direction, prepared complete specifications for the particular conditions required; but I cannot find that any two are alike; and each foundry, in bidding for the supply of such pipe, makes the price to cover the conditions required. I, therefore, have the honor of offering to the Institute a form of specifications for cast-iron coated water-pipe, that may be found useful to water-works companies and others requiring pipe of this kind. The conditions are all reasonable, and can be carried out by makers without additional cost, and with advantage to the purchaser.

Section 1 gives the kind of pipe, with its general characteristics.

Section 2 specifies that neutral pig-iron shall be used, since such iron possesses the strength required, and is uniform in its shrinkage.

Section 5, requiring makers to give the brands of pig-iron used

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in the cupola, is intended to guard against the use of cinder-iron. The characteristics of the ores used in blast-furnaces give to each its reputation, and the furnaces using cinder are all well known.

Section 7, providing that the shell of each pipe shall be measured for thickness throughout its entire length, enables the inspector to determine whether there are thin places near the middle of the pipe, where such imperfections are most likely to occur. An inexpensive instrument, adapted to such measurements, has been devised.

Section 8 provides for a test-bar, from which to determine the transverse strength of the iron. This bar is convenient in form, can be made with very little cost to the maker, and will give the most satisfactory test of the strength of the iron under the conditions to which the pipe will be subjected in service.

Section 11, requiring that all pipe shall be stripped immediately, if taken from the pit before cooling, is very important, for the reason that when pipes are taken from the pit hot, a deposit of loam is frequently left in spots on the outside, and wherever this occurs, the cooling is irregular, and the shrinking strains are unequal. The weakness thus produced, may not show under the subsequent hydraulic test, but may be developed after the pipe has been subjected to rough treatment in railroad and other transportation into the trenches.

Specifications for Cast-Iron Coated Water-Pipe.

SECTION 1. The pipe shall be of the usual kind known as "Hub and Spigot;" each pipe shall be twelve (12) feet in length from the bottom of the hub to the end of the spigot; the form and dimensions of the hub- and spigot-ends shall be as marked on, and shown by drawings, to be furnished or approved by the purchaser.

SEC. 2. The metal shall be of the best quality for the purpose, made from what is commercially known as "neutral" pig-iron, which shall have been made from iron-ores, without the admixture of cinder; and when cast into pipe, the metal shall be tough, and of such density and texture as will permit of its being easily cut and drilled by hand.

SEC. 3. Each pipe shall have cast on the exterior surface of the hub ——— in raised Roman letters, one and one-half (1½) inches long, and shall be further marked in like manner with letters and figures to designate the maker of the pipe, and the year; also, serial numbers shall be cast on each pipe, commencing with a number to be designated by the purchaser.

SEC. 4. The pipe shall be cast in dry-sand moulds, or flasks, placed vertically, and shall be of the several diameters named in the contract. The shell shall be of uniform thickness of metal, smooth and sound, without cold-shuts, lumps, swells, scales, blisters, sand-holes or other imperfections; truly cylindrical; of full diameters; and with interior and exterior surfaces concentric.

SEC. 5. Chemical analyses of the iron used shall be given whenever asked for, and a written daily report shall be furnished to the inspector, giving the brands of pig-iron used in each day's melting.

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SEC. 6. Each ladle of iron from which the pipes are poured, shall be designated by a consecutive number, beginning with each day's work. A daily record of pipes cast shall be kept and furnished to the inspector, and must record the number of the ladle from which each pipe was poured, as well as the number on the pipe.

SEC. 7. The shell shall be of uniform thickness throughout its entire length. Should the inspector find in any one place a decrease in thickness of more than ten (10) per cent. from that specified in the contract, such pipe shall be rejected.

SEC. 8. From each ladle of melted iron there shall be cast one test-bar, not less than twenty-six (26) inches long, two (2) inches wide, and one (1) inch thick. This bar shall be tested for transverse strength when loaded in the center, twenty-four (24) inches between supports (narrow sides vertical), and shall carry a center breaking-load of not less than two thousand (2000) pounds, and show a deflection of not less than $\frac{3}{10}$ inch before breaking. This bar shall be cast as near as possible to the specified dimensions without finishing up, but correction will be made for variation of width and thickness, and the corrected result must conform to the above requirements.

SEC. 9. If any test-bar shows a breaking-strain or deflection less than the above requirements, all pipe made from the corresponding iron shall be rejected.

SEC. 10. The maker shall furnish to the inspector in charge two wrought-iron rings (male and female templates), the one showing the outside diameter of the spigot, and the other the inside diameter of the hub-end of each size of pipe, and the said pipe shall conform in diameters to the respective templates.

SEC. 11. The pipe shall be thoroughly cleaned, both inside and outside, without the use of acid or other liquid. If the pipe is stripped while above a dark-blue heat, all sand must be removed from the outside immediately.

SEC. 12. When the pipe is so cleaned, it shall be heated to 300 degrees Fahrenheit, and immersed in a bath of coal-pitch varnish of an equal temperature. When the pipe is removed from the bath, this coating shall *fume freely*, and set perfectly hard within one hour from the time of removal. No pipe shall be tested by hydrostatic pressure until the varnish has become hard.

SEC. 13. When coated, the pipe shall be subjected to a test by hydrostatic pressure of not less than four hundred and fifty (450) pounds per square inch for 6 inch and 8-inch; four hundred (400) pounds for 10-inch and 12-inch; three hundred and fifty (350) pounds for 16-inch and 20-inch; three hundred (300) pounds for 24-inch and 30-inch; and two hundred and fifty (250) pounds for 36-inch diameter and upwards; and while under such pressure the pipe shall be subjected to an additional test by a series of smart blows at various points throughout its entire length with a three (3) pound hammer attached to a handle sixteen (16) inches long. If any failure is shown in the pipe during this test, it shall be rejected.

SEC. 14. The weight of each pipe must correspond as nearly as possible to the standard named. Any pipe which may fall short over five (5) per cent. of the standard weight shall be rejected, and no weight shall be allowed or paid for on any pipe that shall exceed two (2) per cent. of the standard weight. These requirements shall be determined by the weight of each pipe separately.

SEC. 15. All special castings, such as "Ells," "Tees," "Ys," bends, crosses, etc., shall be made from the same mixture of pig-iron as has been approved for pipe; and all such castings shall be submitted to the same care in cleaning and heating, and the bath shall be used under the same conditions as for straight pipe.

SEC. 16. All appliances necessary for the inspector to carry out the requirements of these specifications shall be furnished by the maker, free of cost to the purchaser.

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Discussion.

J. C. Bayles, East Orange, N. J. (Communication to the Secretary): As a purchaser or user of cast-iron pipe, or any other kind of pipe, I should not be satisfied with so much of Mr. Yardley's specifications as relates to coating and testing. I should in every case demand that the pipe be tested before it is coated. This is especially true of cast-iron. Any form of coating suitable for a pipe which is to be shipped and handled will conceal a multitude of small defects in casting, such as blow-holes, sand-holes, and even small shrinkage-cracks. In 1887 the Commissioners of the Health Department of New York City found it necessary, in the discharge of their responsibilities relative to the plumbing of new buildings, to amend the plumbing-code by the insertion of a clause forbidding the use of dipped, varnished or painted pipes. The reason for this action was the discovery that even a thin coating, of very little value as a protection for the iron, could be made to conceal so many foundry defects, and permitted so much carelessness on the part of makers, that sound pipes were the exception rather than the rule. The value of such a coating in enabling pipes to pass a test is shown in much of the light riveted pipe in the market. I have seen a pipe put together with small cold rivets, headed by pressure, which, after coating with coal-tar and asphalt, has shown in the testing machine a capacity to carry without leaks a test-pressure much higher than Mr. Yardley prescribes for the strongest and heaviest cast-iron pipes he mentions. Before coating the seam would have shown a continuous leak from end to end. The fact that a coated pipe will hold water under a considerable pressure in the testing-machine seems to me to speak more for the coating than for the pipe. A cork inserted in the end of a gun-barrel will usually be found in place when the explosion of the charge has burst the barrel. Experiments show that a plug of mud, or even of snow, will do the same. As a manufacturer of pipes, I should be very much pleased to have consumers specify that none should be tested until they had been coated. We should have very few wasted.

For cast-iron pipes the test-pressures prescribed by Mr. Yardley seem to me much higher than is necessary. The caulked joint is not adapted for high pressures, and cast-iron pipe is seldom employed under pressure greater than a lead caulking will stand. It is not unusual in water-works practice for the lead to blow out of pipe-joints at pressures much below those which the pipe will carry safely and comfortably. This is especially true if opportunity is offered

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for such changes of temperature as will produce a measurable contraction and expansion with the changes of the seasons. The strength of the pipe cannot be greater than the strength of its joints; and with the hub-and-spigot connection the usefulness of cast-iron pipes as a pressure-conduit has well defined limitations. In buying it for any purpose for which that kind of pipe is adapted I should be content with half the test-pressures prescribed by Mr. Yardley, but should insist on having the test made before the pipe was dipped—and, for that matter, I should require a very much better coating, after testing, than a coal-pitch varnish applied at 300° F.

MR. YARDLEY (Communication to the Secretary): Cast-iron pipes for water-works were first used by the New River Water-works Company, of London, England, in the year 1810. In 1822 the greatly increased demand for water made it necessary for the company to enlarge the mains. In taking up some of the pipe laid in 1810, many were found rusted so much that they were unfit for future use. This fact coming to the notice of Dr. Angus Smith, he commenced a series of experiments in the preparation of a coating or varnish to be applied to the inner and outer surfaces of pipe to be used for carrying water. His experiments demonstrated the fact that his varnish would not stop the rust when it had once commenced, notwithstanding the rust was covered by the varnish. It was then he adopted the system of coating pipe as soon after cleaning as was practicable, and requiring the pipe to be heated to 300° F. before immersing it in the bath.

In 1858, some of the pipe being taken from the trenches, in which they had been under ground for thirty years, the varnish was found hard and bright, and the pipes that were broken gave no sign of rust. So satisfactory was this result that the English Government gave a patent therefor to Dr. Angus Smith, and his process has since been used by all makers of cast-iron pipe for water-works.

In making my specifications I have followed what I believe to be the recognized practice of the best hydraulic engineers of this country, and I know I have the endorsement of many.

I was not aware, until Mr. Bayles made the statement, that tar-coating possessed so much strength. If he is correct, would it not be more economical for water-works companies to have the iron shell of the pipe much thinner than now used, and give the pipe greater thickness of tar?

In regard to the hydraulic test, I have the best authority for the

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pressure required for all pipe above 16 inches diameter. The increased pressure is specified for the smaller diameters, because the water-test is the only one used. It would be very inconvenient, as well as quite expensive, for the makers, if they were required to take test-bars for so great a number of pipe as the foundry could turn out daily of the smaller sizes.

THE HEROULT PROCESS OF SMELTING ALUMINUM ALLOYS.

BY F. P. DEWEY, WASHINGTON, D. C.

(Washington Meeting, February, 1890.)

RECENT discussions have furnished to the members of the Institute much information concerning aluminum and its applications. The present paper will be devoted almost entirely to the description of a process by which the metal is produced in the form of alloys.

The characters which especially distinguish the Heroult process consist in electrolyzing a mass of molten alumina (Al_2O_3), and at the same time alloying the separated aluminum with a heavy metal, the heat necessary for the fusion of both the alumina and the alloying metal being supplied by the direct passage through the molten Al_2O_3 of an electric current of high amperage and low voltage.

The process has been in active operation at Neuhausen in Switzerland for about two years. It is also in active operation at Froges in southern France. Both these locations were selected for this purpose on account of the presence of large and desirable water-powers.

The apparatus consists of a carbon containing-vessel, which may be a single crucible for small operations, or may be built up of slabs or blocks of carbon for large and industrial operations. When built up of slabs or blocks, the whole is surrounded by an iron casing. In the earlier crucibles this iron casing was cast around the carbon, in order to secure close electrical contact. This, however, has been found unnecessary; and in the later crucibles the casings consist of plates bolted together, while pulverized carbon is packed between the carbon blocks and the iron. Electric conductors are united direct to this iron casing, and the current passes from this through the carbon lining to the interior of the crucible. By this

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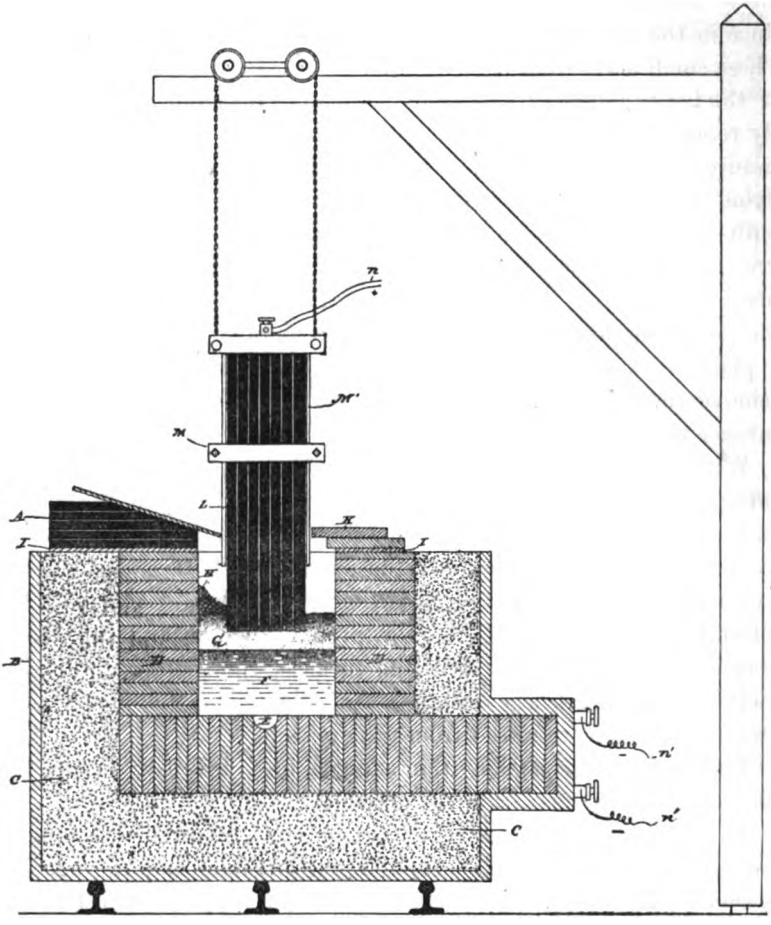
arrangement the crucible forms a part of the electric circuit, and in the earlier stages of the process is the cathode. When, however, the metal within the crucible melts, the melted metal serves as the cathode during the running of the process.



THE HEROULT FURNACE FOR THE PRODUCTION OF ALUMINUM ALLOYS, BY THE ELECTROLYSIS OF MOLTEN ALUMINA.

The accompanying illustration shows the general construction of the Heroult furnace. In this figure the different parts are lettered as follows: A, charging-opening; B, iron casing; C, pulverized carbon packing; D, carbon plates; E, tapping-hole; F, molten metal; G, molten alumina; H, solid alumina; I, alumina grout;

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K, cover-plates; L, anode; M, clamp; M′, metallic side-plate; *n*, positive conductor; *n′*, *n′*, negative conductors.

A single carbon rod may serve for the positive electrode or anode in small operations. For large and industrial operations it was formerly necessary to build up compound anodes of slabs of carbon, since in the earlier days it was impossible to get large single slabs. These small slabs were held together by dowels, either of carbon or of the heavy metal of the alloy, and clamped together at the top. By recent work at Froges it has, however, been possible to manufacture single pieces of carbon of any desired size to serve as anodes. From these very much better results are obtained than from the built-up anodes. One of the great difficulties with the latter is that, even with the utmost care in their preparation, and in testing the plates before use, they will crack during the operation, and the pieces, falling down, are apt to cause short circuits. Moreover, when a plate cracks, it becomes necessary to interrupt the operation and remove the broken piece. On account of the high heat in the furnace this is a disagreeable operation.

With all the large anodes, whether built up or solid, metallic side-plates are used. These are clamped tightly upon the anode and extend a short distance into the furnace when new. They serve to protect the heated portion of the anode outside the furnace from oxidation, and they conduct a portion of the current, and thereby lessen the outside heating. As the process goes on and the anode is consumed and lowered into the furnace, these plates gradually melt and furnish heavy metal to the alloy. As the plates melt away the clamps are raised from time to time.

The anode is suspended from a crane provided with means for its ready adjustment within the crucible.

In starting the process in a cold crucible, a suitable amount of copper is placed upon the bottom of the crucible, and the current is then allowed to pass. This soon fuses the copper, which forms a bath of molten metal, and thereupon becomes the cathode. Alumina is now added and the anode is slightly raised. In the intense heat thus produced the Al_2O_3 melts and forms a bath above the melted copper. The melted Al_2O_3 between the carbon anode and melted metal cathode becomes an electrolyte. The current passes through it and it is decomposed, the aluminum separates at the cathode and alloys with the heavy metal, while the oxygen goes to the anode and combines with the carbon. It will thus be seen that the process is electrolytic in its operation.

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In the ordinary operation of the process the distance between the electrodes is maintained at about 3 mm. At a suitable point in the circuit is placed a large ampere-meter, by watching which the work man can judge as to the distance between the electrodes, and raise or lower the anode as required.

Alumina and copper are added from time to time as the operation of the process requires, and the reduction is continued until the crucible is full of metal, when it is tapped into a casting-ladle, and the current is stopped.

A considerable portion of melted alumina always flows out of the crucible after the alloy. This is returned to the furnace during the next run. As soon as the crucible is empty, the tap-hole is stopped and the proper amount of copper is added. This is melted by the residual heat of the crucible. The current is again turned on, alumina is added, and the operation of reduction is resumed. From ten to fifteen minutes are required for this work. As soon as the metal in the ladle has cooled sufficiently it is cast in iron moulds. This cycle of operations is repeated so as to constitute a continuous process.

Nothing but copper and alumina are charged into the furnace. As the copper and aluminum are withdrawn in the form of an alloy and the oxygen passes off as gas, there is no slag or refuse of any kind arising from the operation.

As the anode is consumed in the process it is gradually lowered, and when only an unusable stump remains, it is hoisted out of the furnace and a new one is introduced. This is usually done at tapping-time, and takes from twenty to thirty minutes. With the built-up anodes formerly used, it was necessary to heat up the new anodes gradually, and generally an hour was consumed; but with the new block-anodes, the current can be turned on at once.

The readings of the ampere-meter give an excellent guide at all times as to what is going on within the crucible. When everything is moving along smoothly and properly, there is simply a slight oscillation of the needle. Sudden or wide variations indicate some irregularity in the operation.

In irregular work the position of the anode with reference to the cathode and the melted alumina is clearly shown. If the current passes between the melted alumina and the anode by an arc, if the anode dips into the alumina, but not far enough, or if too far, each condition will be shown by the position of the needle upon the circle; while if by any means the anode comes within arc-distance of

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the melted metal at the bottom of the crucible, the fact will be unmistakably indicated upon the ampere-meter by a sudden increase in the number of amperes generated. This last condition is also clearly shown by a number of other circumstances. When copper-alloys are made, there will be a copious volatilization of the metal, and the flame at the charging-opening will be strongly tinged by it.

The proper operation of the process requires that the distance between the electrodes be maintained as uniformly as possible. The distance found most suitable under ordinary circumstances is about 3 mm. By watching the ampere-meter the workman can judge as to the distance and raise or lower the anode as required. Generally he has only to lower the anode gradually as it is consumed.

The proportion to be observed between the alumina and copper added varies under different circumstances. It is designed to produce at one operation an alloy carrying a high percentage of aluminum. Alloys as high as 73 per cent. of aluminum have been produced; but under certain circumstances it is more economical to run from 35 to 40 per cent., and again at times it is more desirable to produce a 25 per cent. alloy. All things being considered, the 25 per cent. alloy is probably the most economical to produce, and may be taken as the standard.

With proper care, and barring accidents, the grade of metal produced can be held with much uniformity. The maximum variations need not exceed 1 per cent. of aluminum.

The percentage of aluminum that can be introduced into the alloy by this process is governed by the question of specific gravity. Since the specific gravity of alumina is 3.7 to 4.1, and the specific gravity of aluminum only 2.6, the limit in the percentage of aluminum is reached when the product becomes light enough to float upon the melted alumina.

Practical work has shown that when the copper alloy reaches about 65 per cent. of aluminum, it begins to rise and cause trouble by short-circuiting between the sides of the crucible and the anode. When 73 per cent. aluminum is reached, there is so much short-circuiting from this cause that the process must stop.

The loss of material varies with the percentage of aluminum introduced into the alloys. In making a 20 per cent. bronze the loss of copper is from 5 to 6 per cent., but in making 30 per cent. bronze, it may rise to 10 per cent. Owing to the high temperature and the rapid evolution of gas at the anode, there is considerable loss in alumina carried off mechanically by the gases, which may amount

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to 30 per cent., but with suitable means for recovering the solid material in the fume this has been greatly reduced at the Froges works. When the process is in smooth working-condition, the flame and smoke are alike light-colored. When, however, a short circuit occurs, the flame becomes tinged with copper and is very much darkened.

Other metals besides copper can be used to alloy with the reduced aluminum, the only limitation being that they shall not be volatile at the temperature of the operation.

The rich alloys produced in the furnace are very hard and brittle. They are not fit for any direct application. To prepare the useful alloys, these rich products are diluted to the proper degree by the appropriate metal for each case.

While this process is more particularly designed for the reduction of aluminum, yet it is also adapted to the reduction of other refractory metals. Silicon-copper has been thus prepared upon an extensive scale.

Since the current required is different from that used for any other purpose, in that it must be of large quantity and of low intensity, the dynamo has to be especially built for this purpose. Machines of such abnormal current have never before been called for, and when their output is compared with the low speed at which they run, they may well be classed with the largest dynamos which have ever been built.

The construction of these gigantic machines was undertaken by the Oerlikon Machine Works, Zurich, Switzerland; and Mr. C. E. L. Brown's construction of the armature, in which the winding is passed through holes in the core, is particularly adapted to this application. A considerable number of standard 150 electrical horse-power machines have already been made at these works.

The field of the dynamo is of the multipolar type, having 6 poles. It is made in one casting, in order to avoid all joints, and weighs 10,000 kilogrammes. The exciting of the field-magnets is effected by a separate machine.

The armature is of the usual Brown type, with embedded wires, but contains two distinct circuits, each having its own commutator. The winding consists on the outside of round copper bars, and on the inside of flat copper plates, placed in grooves planed out of the wood lining, so that the plates are firmly held, and at the same time insulated.

There are six sets of brushes, making in all 72. Although cross-

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connections between the equi-potential coils are not absolutely necessary, yet they have been provided. Copper bars conduct the electric current from the machines to the furnace.

The two machines originally built for the works at Neuhausen were connected direct to a Jonval turbine. They were designed to give a current of 6000 amperes at 20 volts, when running at 180 revolutions. It was found in practice, however, that they could be worked up to 30 volts. The present plant at Neuhausen consists of two 600 horse-power machines running on vertical shafts, and one 300 horse-power machine.

The process has been in operation at Boonton, New Jersey, for a short time. The installation here consists of a Victor 32½-inch turbine-wheel, supplied with water through a 6½-foot flume, with a fall of 28 feet. This, at 220 revolutions, is designed to give 220 horse-power. Coupled direct with the turbine is a Brown dynamo of 150 E. H. P., which furnishes a current of 3500 amperes at 33 volts. A small 15 horse-power exciting dynamo is used with the large machine.

There are two furnaces, one 43 inches square on the outside, built up of carbon plates, giving a crucible of 9 inches by 16 inches, and 19 inches deep. This crucible has a capacity of 200 lbs. of contained aluminum in twenty-four hours, after making due allowance for stoppages at casting, changing anodes, and for minor repairs to the crucible. The small crucible is for use in determining the applicability of various raw materials available in this country.

The anode first used consisted of ten plates, 6 x 12 x 40 inches, bound together at the top by a specially designed clamp, to which the electric conductors are attached. Upon the outside of the anode were placed plates of copper clamped at their lower ends firmly against the carbon plates. At present the solid block anodes are used exclusively.

In preparing the crucibles for work a layer of alumina is spread over the top, and upon this slabs of carbon are placed to form the cover, leaving just sufficient opening in the centre to permit the easy movement of the anode. Upon the front an opening is left through which the materials are charged. The grout of alumina serves to insulate the cover from the body of the crucible.

The current is brought to the crucible through four large copper bars, no particular care being taken to insulate them, since a small loss in amperes will not much affect the working of the process.

For raw materials the purest artificial alumina and Lake copper

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are used in the regular running of the furnace. Of these, about 25 lbs. alumina and 30 lbs. copper are charged at intervals every hour. Two to three castings are made during the twenty-four hours.

Early in March a test-run was made to determine the general efficiency of the plant. This run lasted 12¼ days. During that time 10,000 pounds of bronze, averaging 22.1 per cent. aluminum, were produced. The total loss on the copper was only 6.4 per cent.

Each casting of this run has been analyzed, showing a metal of remarkable purity. The following are representative analyses:

Rich Aluminum Bronze.

Run.	Al.	Cu.	Si.	Fe.
23	28.41	71.16	0.10	0.33
30	26.35	73.25	0.11	0.29
36	24.83	74.80	0.11	0.26
37	26.66	72.95	0.12	0.26
49	19.65	80.15	0.14	0.07
51	24.95	74.60	0.12	0.33

Analyst, Dr. F. M. Simonds.

A considerable portion of the impurity consists of iron introduced from the tools used at tapping. The tap-hole is usually very hard, but by an improved method of tapping it is hoped to reduce the percentage of iron in the product.

One of the chief objects in setting up this small plant was to determine the applicability of a number of American raw materials for this process; and a considerable portion of the time is occupied in this work of testing.

The process is protected by a number of patents, both in the United States and abroad.

The following series of tests, made by Professor Tetmeyer, of Zurich, on metals produced at the Swiss works, shows the general character of the merchantable alloys prepared from the product of this process:

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Metal.	Ultimate Stress. Kg. per sq. mm.	Extension. Per cent.
Aluminum bronze, A	34.6	25.4
" " B	38.4	27.4
" " C	45.0	44.5
" " D	45.7	35.0
" " E	51.0	35.0
" " F	53.3	20.5
" " G	57.4	14.5
" " H	61.0	7.8
Aluminum brass.....	48.1	20.7

[NOTE BY THE SECRETARY.—It is claimed by Mr. Eugene H. Cowles, Cleveland, Ohio, that the process described in the foregoing paper is an infringement upon the Cowles process (*Trans.* xiv., 492). While Mr. Dewey entertains a different opinion, it is but just to both parties to say that they do not desire to introduce into the *Transactions* any discussion of the controversy. The foregoing paper was offered simply as a description of actual practice, personally observed by the author, and without reference to any disputed question of novelty, priority or proprietorship. Under these circumstances it is proper to put on record, for the information of readers of the paper, the fact that such a question has been raised, although the investigation and decision of it must be made elsewhere.]

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*THE LEWIS AND BARTLETT BAG-PROCESS OF COLLECTING LEAD-FUMES AT THE LONE ELM WORKS, JOPLIN, MISSOURI.**

BY F. P. DEWEY, WASHINGTON, D. C.

(Washington Meeting, February, 1890.)

THE most serious problem that confronts the lead-smelter is the waste caused during smelting by the volatilization of both lead and silver, which are consequently lost in the form of fume. It is not difficult to collect a portion of the solid materials passing out of the

* Presented by the kind permission of Professor G. B. Goode, Assistant Secretary, Smithsonian Institution, in charge of the United States National Museum.

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throat of the furnace, especially the coarser particles that have been carried forward mechanically by the blast, but to remove all the finest particles is a very difficult undertaking.

The use of the fume from lead-smelting, as a paint, is very old, having been mentioned by Bishop Watson in his scientific writings in 1778. In this case the ordinary gray fume was used, but its color was a great objection.

Percy* suggests the use of fabrics for filtering out the solid matter of fumes, but makes no mention of any attempt to do so.

Of all the processes proposed for saving the fumes arising from the volatilization of the metals, one of the most efficient is the Lewis and Bartlett process of collecting in bags. The same principle is applied in collecting oxide of zinc for use as a paint, and the process was originally proposed as a paint-process. Early experiments, however, failed to volatilize so large a portion of the metal in lead-ore as is done with zinc-ore, while, at the same time, it was found impracticable to avoid reduction and consequent formation of metallic lead in the furnace-chamber.

It became necessary, therefore, to modify the process, or rather to combine it with a smelting-process, in order to take care of the metallic lead produced. In continuing the development of the process, it has been proposed to use it to collect the fumes from lead-furnaces generally.

Its chief application, however, has been in the smelting of non-argentiferous lead-ore, where the final fume is obtained as an impalpable white powder, which finds application in several ways, especially as a paint and for a weighting material. This process is fully illustrated in the United States National Museum by a collection of thirty-three specimens, made by E. B. Kirby, E. M., in October, 1884, at the Lone Elm works.

Mr. Kirby also furnished a report upon the process to which I am indebted for many of the details of the operation, and also for the drawings given in this paper.

The works of the Lone Elm Mining and Smelting Company at Joplin, Jasper county, Missouri, are located 1½ miles northwest of the town, upon one of the tracks of the St. Louis and San Francisco railroad. They are owned and operated by Mr. E. R. Moffet and his son, Mr. O. R. Moffet. At the time the collection was made, little lead-ore was being mined in the district. All the other smelters had very much curtailed their operations on account of the low

* Percy, *Lead*, p. 449.

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price of the metal. The manufacture of white paint as a by-product, however, enabled the Lone Elm works to continue smelting.

The ores* of this region occur in the Archimedes or Keokuk limestone of the Subcarboniferous, and vary from a very pure lead-ore through every gradation of mixture to a very pure zinc-ore. They are now almost universally sulphide-ores, the amount of oxidized ores being quite small. Formerly, oxidized lead-ores were much more abundant.

The following analyses† of the prepared galena-ore by Professor Regis Chauvenet show the general character of the ores of the region:

	1	2	3	4	5	6	7	8	9
Insoluble matter.....	0.05	0.61	0.12	.....	0.44	0.71	0.53	0.33	1.72
Lead.....	.....	84.06	85.84	.....	.....	.....	.....	.....	.....
Zinc.....	1.32	0 94	0.12	.....	0.91	0.73	0.52	Trace	0.94
Iron.....	0.09	0.16	0.05	.....	0.45	0.196	0.196	Trace	0.24
Antimony.....	None	None	None	.....	None	None	None	None	None
Silver, oz. per ton.....	1	1¼	1¼	1¾	1¼	¾	¾	¾	1
Sulphide of lead.....	.....	97.05	.....	.....	.....	.....	.....	.....	.....
Sulphide of zinc.....	.....	1.41	.....	.....	1.36	1.08	0.77	.....	1.41
Bisulphide of iron.....	.....	0.34	.....	.....	0.96	0.42	0.42	.....	0.53

No. 1.—Village Diggings, Granby.

No. 2.—Holman Diggings, Granby.

No. 3.—Trent Diggings, Granby. Galena found imbedded in red clay.

No. 4.—Eastpoint, Granby. Fine-grained structure, and light steel-gray color. Supposed to be rich in silver. It contains, indeed, a little more silver than the other sample.

No. 5.—Temple Diggings, Joplin.

No. 6.—Swindle Diggings, Joplin.

No. 7.—Lower Joplin Valley Diggings.

No. 8.—Stevens' Diggings (Turkey Creek district).

No. 9.—Oronogo.

The following analyses were made by Professor C. P. Williams,‡ at the Rolla School of Mines, to determine the small amounts of other metals, besides lead, present in the ores.

* For the geology of the region, with descriptions of the occurrences of the ores, see *Geol. Surv. of Mo.*, 1873–74, pp. 384–502, by Schmidt & Leonhard; also, article by F. L. Clerc, in a pamphlet, published by the Joplin Board of Trade, entitled "Lead and Zinc Ore of Southwestern Missouri."

† *Geol. Surv. of Mo.*, 1873–4, p. 388.

‡ *Trans.*, v., 315.

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	Antimony.	Copper.	Iron.	Zinc.	Nickel.
I.....	Trace.	Trace.	.05867	.06782	.....
II.....	.07429	.00478	.02169	.00938	.0660
III.....	.00551	.00239	.08602	1.35554	.....
IV.....	.02764	.01677	.03220	1.76558	.....
V.....	Trace.	None.	.12040	.14679	.....

I.—Washed galenite, from jig, at Granby, Newton county, Mo.
II.—Block mineral, Joplin, Jasper county, Mo.
III.—" No. 1," block mineral, Granby, Newton county, Mo.
IV.—Holman's Diggings, Newton county, Mo.
V.—Birch Diggings, Joplin, Jasper county, Mo.

The supply of ores for Lone Elm works came chiefly from the shallow surface lead-mines at Galena, Kansas, and from a few mines in the south part of Joplin.

The ore consists of galena, associated more or less with blende. It is crushed and jigged by hand at the mines, steam-jigs and sizing-apparatus being employed at the larger mines. The dressed galena, known as " mineral," is bought on inspection, and without assay. It was worth, at that time, about $19 per 1000 pounds. The price rises and falls with the price of pig-lead in St. Louis. When lead is worth 4 cents per pound in St. Louis, 70-per-cent. mineral is worth $24 per 1000 pounds. A difference of ¼ cent per pound in the price of lead represents $2 per 1000 pounds in the mineral. First-class mineral contains from 70 to 73 per cent. of lead. More blende is generally present than is indicated by the analyses quoted above.

The usual method of smelting in the region is by American-Scotch hearths, the rich slags produced being afterwards reduced in a low shaft-furnace or " slag-eye," with two tuyeres. The furnaces used at the Lone Elm works, are modifications of these two forms, and are designed to produce a large amount of fume.

The works were started in 1868, smelting with Scotch hearths. In 1878 the manufacture of white paint began, the fumes from the furnaces being collected in bag-rooms constructed under the patents of E. O. Bartlett and G. T. Lewis, of Philadelphia. Seven hundred and twelve tons of crude white lead were shipped that year, and a more or less regular production has been maintained ever since; but it was only after a long series of costly experiments that the process was brought to its present completeness, and the white fume to a desirable quality.

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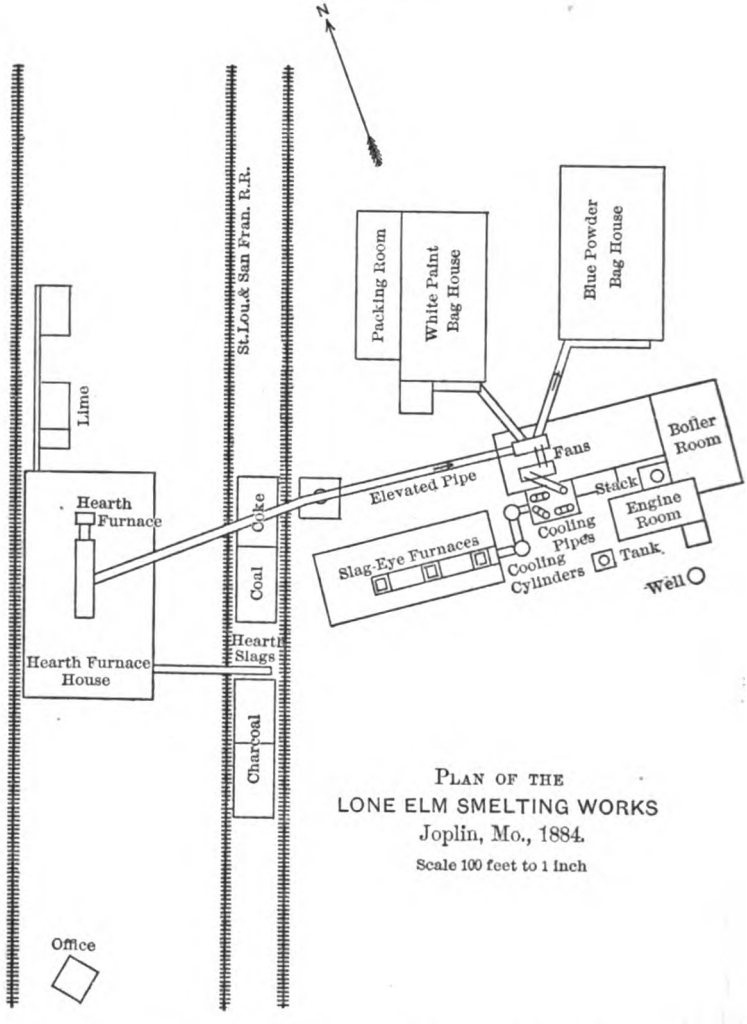
The galena is first smelted with raw coal and slacked lime in a furnace, which is simply an enlarged Scotch hearth. By using a hot fire it is possible to treat a very large quantity of mineral. The hotter the fire the more fume is produced. The products of this

FIG. 1.



PLAN OF THE
LONE ELM SMELTING WORKS
Joplin, Mo., 1884.
Scale 100 feet to 1 inch

smelting are pig-lead, pasty slags rich in lead and zinc, and the fume. The latter is drawn off by an exhaust-fan, through a settling-chamber and a long iron flue, to a bag-house, which contains a great number of woollen bags for filtering the fume out of the gases. This

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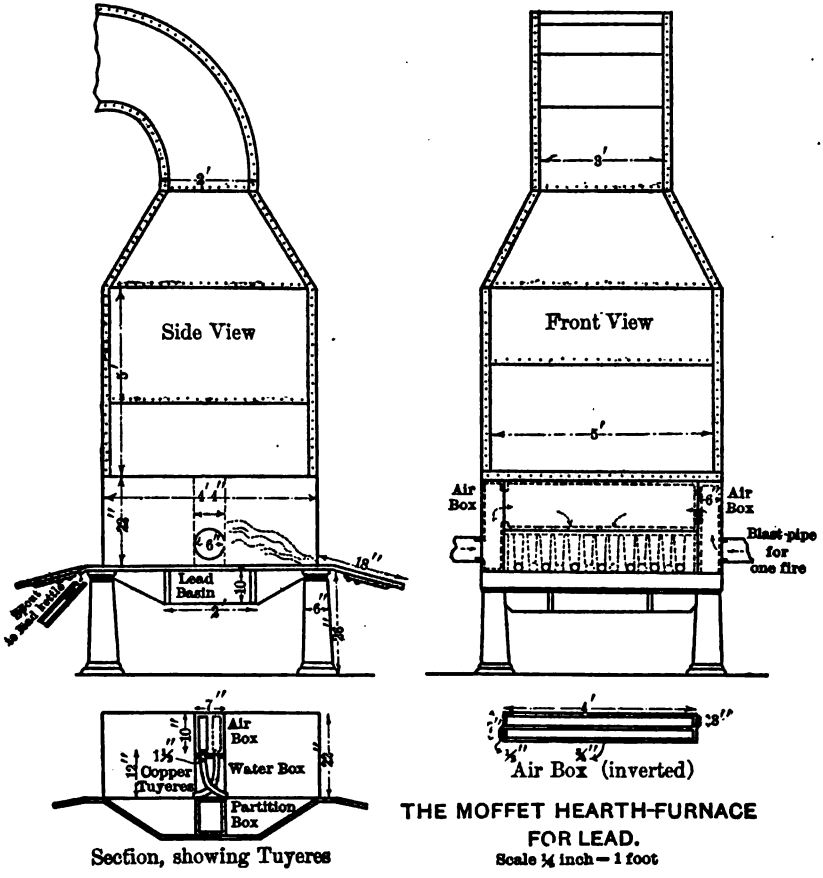
fume is a lead-colored impalpable powder, known as "blue powder." It is ignited and allowed to burn for several hours, by which means it is converted into white coherent crusts. These crusts, together with the hearth-slags and some oxidized ores, are next charged into a modified slag-eye furnace with a very hot and bright coke fire.

The products of this smelting are pig-lead, slags poor enough in

FIG. 2.



THE MOFFET HEARTH-FURNACE FOR LEAD.
Scale ¼ inch = 1 foot

lead to be thrown away, and the fume. In this case the fume, being perfectly white and in a fine state of subdivision, is very suitable for a white paint, for which purpose it finds ready sale.

The ground plan of the works is shown in Fig. 1. The large house, shown in this plan, in which the hearth-furnace is placed,

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was originally built for 16 Scotch hearths, which explains its present shape.

An elevated railroad track brings in mineral, coal and lime on one side of the house, and an elevated wagon road allows mineral to be delivered to the other. The furnace is at one end of the building and the dust-chamber along the center, while a row of bins occupies each side.

A track runs around the room and leads out in the yard to the railroad track, and a hand-crane with a swinging jib is placed near the door for handling pig-lead.

As the mineral is shovelled from the cars it falls through a chute to a pair of rolls, where it is crushed to bean- or nut-size, and falls upon the main floor below.

The rolls, 12 inches in diameter and 16 inches face, are set $\frac{1}{4}$ inch apart. There are two sets on each side of the room, driven by a small vertical engine of 10-inch cylinder, 10-inch stroke, and running at 100 revolutions, with 70 pounds of steam.

The hearth-furnace known as "Jumbo" is upon a wooden platform at the end of the dust-chamber. Its construction is shown in Fig. 2. A simple iron trough, 4 feet long, $3\frac{1}{2}$ feet wide, and 10 inches deep, is divided by a hollow partition along its middle, and supported 26 inches above the floor by four legs. Upon each end of the trough there is a water-box 6 inches wide and 20 inches high. The partition along the center is also carried up 20 inches by two long hollow boxes. The upper one receives the blast, which passes from it by copper tuyeres. These tuyeres pass downward through the lower box, which is filled with water, and issue from its side at the level of the top of the trough. There are 14 tuyeres, 7 on a side. The nozzles are 1 inch in diameter.

The furnace thus comprises two independent hearths, one on each side. The trough is always full of melted lead, which is kept at the same level by an overflow at one corner. The fire is upon the lead, the burning fuel being heaped about 10 inches high along the row of tuyeres. The hottest part of the fire is surrounded by the water-boxes. The air-box above warms the blast. There is an iron apron or shelf along the front of each trough to retain the materials of the charge, and upon which the fragments of slag are picked out. Above the trough the furnace is enclosed in a sheet-iron hood $4\frac{1}{2}$ by 5 feet, and 11 feet high. This is connected with the dust-chamber by a square iron flue about 15 feet long. Along the front of each fire there is an opening in the hood 15 inches high.

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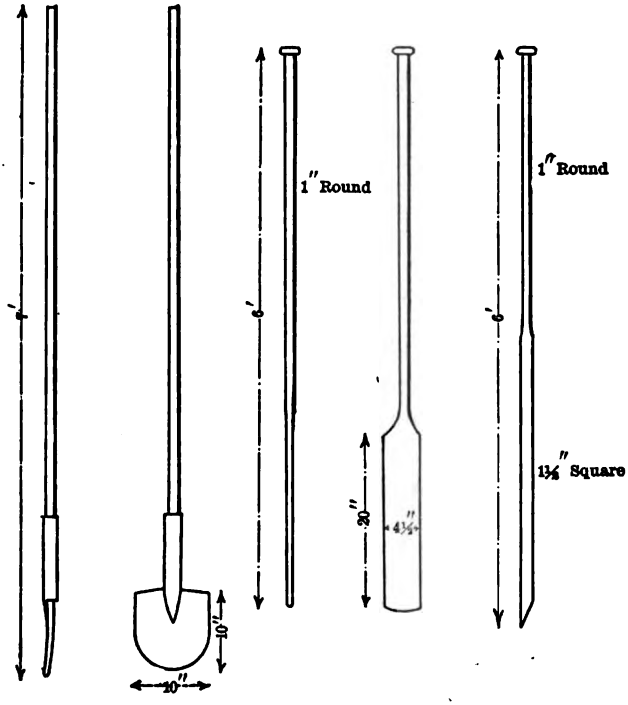
The furnace stands on a platform 30 inches above the floor-level. This affords a working space 15 feet wide and 12 feet long in front of each fire. Inclines from the floor allow the barrows of material to be wheeled up to the furnace. On the other side of the platform are two water-pans and bins for the slag, which is picked out of the fire in pasty, red-hot lumps, and tossed into the water. After cooling, the lumps are shovelled out into the bins below. Each of the

FIG. 3.



Tools used at Hearth Furnace, Lone Elm.

pans is enclosed in a hood for carrying off the steam. There is by each fire a coal box, and an iron box 2 feet square for slacked lime.

The fume is drawn off into a settling-chamber, where most of the dust or mechanically-transported particles, including nearly all of the coal and lime carried over, is retained.

The lead overflows from the hearth into a casting-pot, placed in a pit at one side. This pot, 31 inches in diameter and 44 inches deep, is set in a cast-iron cylinder, and kept hot by a fire of wood. About one cord a week is required for this purpose.

The metal is drawn out by a pipe and stop cock. The stream of

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lead is directed by a short, movable spout into the moulds, which stand on a truck upon a track below the tap.

The pigs, which average 84 pounds, are loaded on a small hand-truck holding 20, and are run upon a track to the scales, and thence to the cars. One man attends to this work.

Raw coal, of rather poor quality, is used. It is mined on the lands of the company at Carbon, Kansas, and brought by rail, costing at the works 7½ cents per bushel of 80 pounds. It is very rusty and contains much sulphur and ash, but is good enough for this purpose. It is also used under the boilers. It contains:

Coal, No. 55,162.

	Per cent.
Water,	2.36
Volatile matter,	34.50
Fixed carbon,	59.37
Ash (pink and white),	3.77
	100.00
Sulphur,	2.42

The lime comes from a kiln a few miles from Joplin, and costs 20 cents per bushel of 60 pounds. It is stored and slacked as required in a small building adjoining. It is used in the furnace in a moist state.

The sample of lime was not analyzed until some time after it was taken, when it yielded the following figures:

Quicklime, No. 55,163.

	As received. Per cent.	Calculated to quicklime. Per cent.
Loss on ignition,	30.07	
Insoluble residue,	1.52	2.16
$(FeAl)_2O_3$,	0.28	0.40
Lime,	68.07	96.92
Magnesia,	0.36	0.51
	100.30	99.99

The prepared mineral is dumped on the floor in front of the fire, and mixed with 1.5 to 2 per cent. of lime. Each fire is worked by two men in 8-hour shifts, so that there are 12 smelters for the two fires. The course of the operation is as follows: The fire remains undisturbed for one and one-half minutes after the last charge, and by this time has become somewhat crusted on top. One man then pries up the fire to loosen it, using the bar shown in Fig. 3. He

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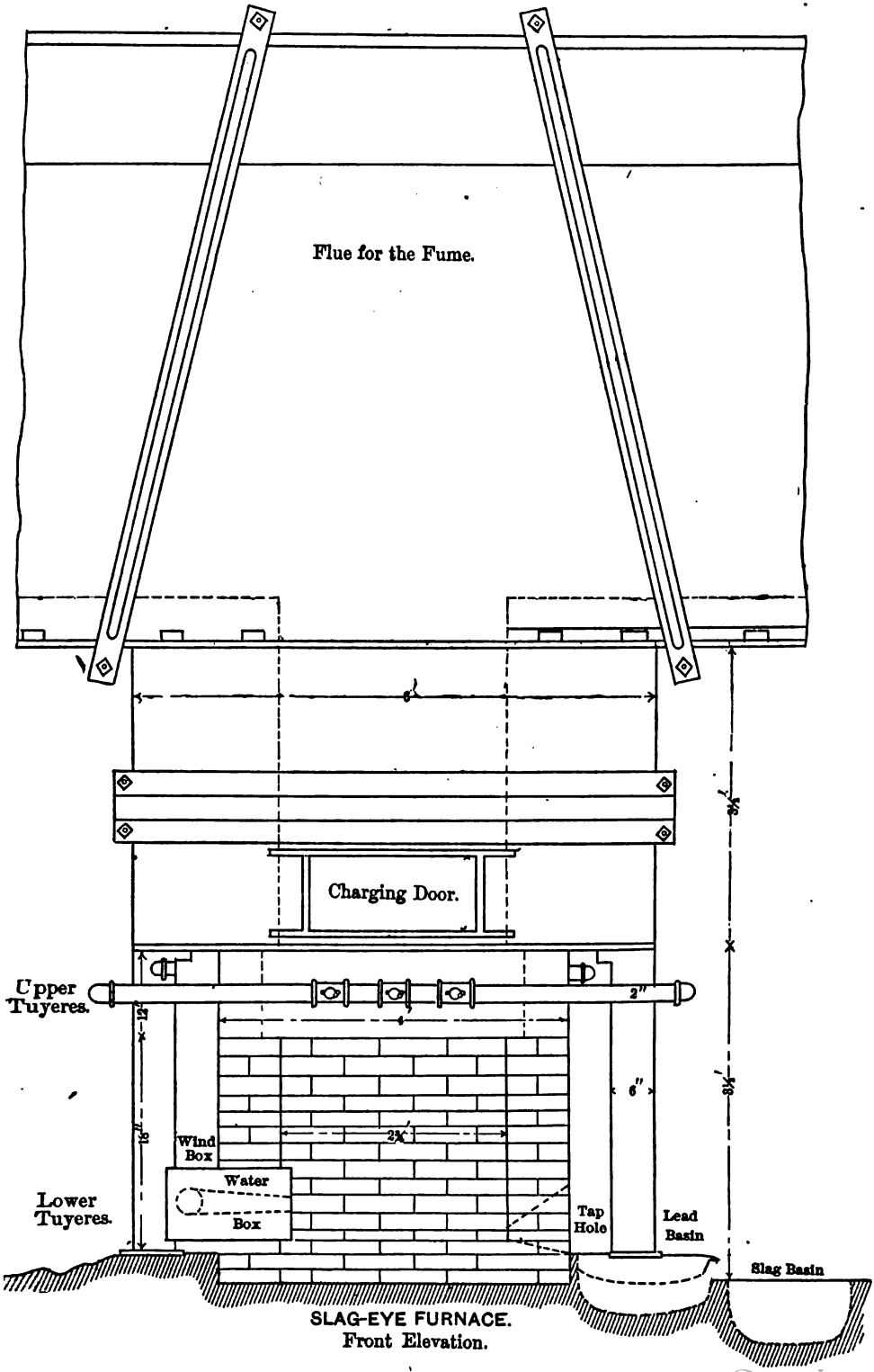
FIG. 4.



SLAG-EYE FURNACE.
Front Elevation.

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plunges the bar into the melted lead beneath, and, using the edge of the trough as a fulcrum, lifts up the mass. In this way he begins at one end of the fire and proceeds to the other; while the second man, with the long-handled shovel shown in Fig. 3, picks out the lumps of the glowing, fuming "slag." With this shovel the "slag" is taken out on the apron, and then tossed into the water-box. The lumps are easily recognized in the fire.

When the men have passed from one side of the fire to the other, the second man exchanges his shovel for the broad paddle, shown in Fig. 3, and loosens the fire a second time. He uses the paddle in the same way as the bar, and is followed from one side to the other by the first man, who picks out the slag with the shovel. The paddle is then laid aside, and both men use the shovel until most of the slag is removed. A couple of shovelfuls of coal are scattered over the fire, and then five or six shovelfuls of the mixture of mineral and lime.

The whole operation has taken two minutes, and the men then rest for one and a half minutes, during which time the fire heats up again, and the mineral over its surface oxidizes. Part of the lead is reduced and sinks through the fire, while part passes off as fume, and the remainder goes into the slag. After one and a half minutes the fire is worked again in the same way as before.

When the tools are not in use they rest conveniently with one end on the apron, and the other in a swinging hook suspended from above. The mode of working the fire is much the same as with the American-Scotch hearth; but the fire is kept much hotter, and more fume is thereby produced, while a considerably greater quantity of material is treated.

The action of the fire upon the mineral is very rapid, and the capacity of the furnace is great, but the work is somewhat severe upon the men, and their endurance is the chief limit to the amount treated in a day.

In 1884, during summer weather, 48,000 pounds of mineral were smelted per day, yielding about 24,000 pounds of pig-lead. The mineral probably contained 73 per cent. of lead, so that 68 per cent. of the lead in the charge was converted at once to metal, and the remaining 32 per cent. to fume and slags, the approximate proportions of these being 2.8 of slag to 1 of fume. Two bushels, or 160 pounds, of coal, and 40 pounds of lime, were consumed per ton of mineral.

In the winter the furnace is run at the rate of 60,000 pounds, and

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for single shifts it has been run at the rate of 76,800 pounds in twenty-four hours.

The "slag" is a pasty agglomeration of oxidized compounds of lead and zinc with the lime, the chert from the mineral, and some of the undecomposed galena and blende. Most of the zinc, which is always present in considerable quantity in the ores, goes with the "slag."

Naturally, more or less metallic lead adheres to the lumps of slag. The sample forwarded by Mr. Kirby for analysis weighed 2352 grammes. It contained two large pieces and one small piece of lead, weighing 307 grammes or 13.05 per cent. This sample was coarsely crushed, thoroughly mixed, and 600 grammes were passed through a ¼-inch mesh sieve. This yielded 16.69 grammes lead, or 2.78 per cent. Of this sample, 150 grammes were passed through a No. 20 sieve, yielding 5.968 grammes lead, or 3.97 per cent. Of this sample, 30 grammes were passed through a No. 60 sieve, and yielded 0.065 gramme of lead, or 0.22 per cent.; making a total of 19.01 per cent. of metallic lead removed in preparing the final sample. Probably considerable lead remained in the sample in a finely-divided state, but no attempt was made to determine this amount.

The analysis of this material presented unusual difficulties. Several attempts were made upon it without entirely satisfactory results. The figures given are the results of a very careful examination, and are given as obtained, without insisting too much that they represent the actual arrangement of the several bodies. It should be noted that in every case, after removing the sulphate of lead ($PbSO_4$) by $NaHCO_3$, dilute acetic acid completely decomposed the silicate of lead, and retained all the separated silica in solution. A small amount of coal remained adhering to the slag, but it was not submitted to combustion.

Scotch-Hearth Slag, No. 55,175.

Residue.

	Per cent.	Per cent.
SiO_2	1.97	
$PbSO_4$	0.24	
Fe_2O_3	1.67	
Al_2O_3	0.21	
ZnO	0.57	4.66
$PbSO_4$		4 94

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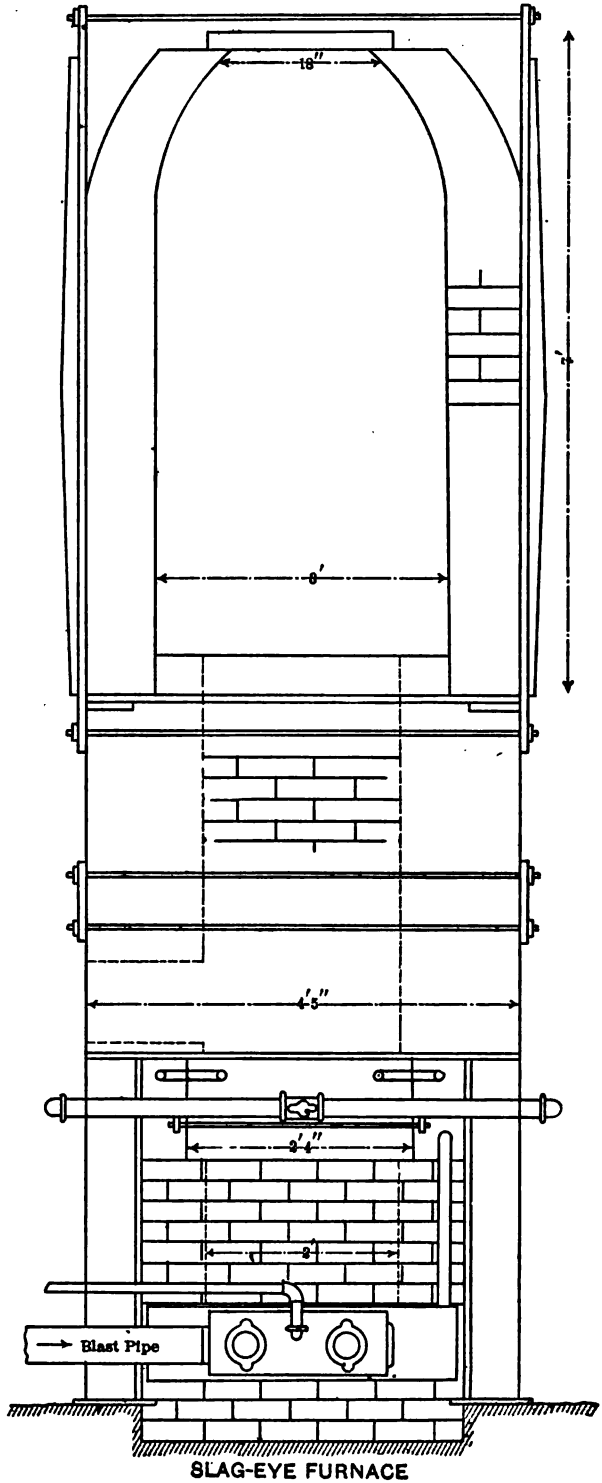
FIG. 5.



SLAG-EYE FURNACE
Side Elevation.

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Acetic Acid Solution.

SiO_2,	10.73	
PbO,	33.55	
Fe_2O_3,	1.23	
Al_2O_3,	0.57	
ZnO,	13.96	
CaO,	11.49	
MgO,	0.12	71.65

Nitric Acid Solution.

PbS,	14.73	
FeS_2,	0.67	
ZnS,	3.64	19.04
		100.29

SO_2 was not present in weighable amounts.

The pig-lead produced in this smelting is of the following composition:

Pig-Lead, No. 55,172.

	Per cent.
As,	0.00011
Sb,	0.00146
Ag,	0.00056
Cu,	0.01782
Bi,	none.
Cd,	none.
Ni,	0.00077
Co,	0.00005
Fe,	0.00686
Zn,	0.00033

The dust-chamber is of brick, 40 feet long, 19 feet high, and 6½ feet wide, with a door in one side for removing the accumulations.

The sample of chamber-dust consists of a coarse-grained powder of mingled light- and dark-colored grains. Particles of coal, and an occasional grain of PbS, can be distinctly seen. Dried at 100° C., it gave:

Dust from Chamber, No. 55,164.

Residue.

	Per cent.	Per cent.
SiO_2,	2.76	
$PbSO_4$,	3.29	
Fe_2O_3,	.43	
Al_2O_3,	trace	6.48
$PbSO_4$,		19.45

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Acetic Acid Solution.

PbO,	3.79	
Fe_2O_3,	0.50	
Al_2O_3,	0.06	
ZnO,	0.35	
CaO,	6.40	
MgO,	0.03	
CO_2,	3.83	
SO_2,	.86	15.82

Nitric Acid Solution.

PbS,	51.59	
ZnS,	5.57	57.16
		98.91

On combustion it yielded:

	Per cent.
CO_2,	14.41
H_2O,	3.38

A partial analysis by Mr. J. A. Allen of a second sample yielded:

	Per cent.
Pb,	51.86
ZnO,	7.41
Fe_2O_3,	1.28
Al_2O_3,	0.41
CaO,	14.01
MgO,	0.16
S,	9.48
SO_3,	5.10

A sample of the dust deposited half-way between the hearth and fan was collected. It was much finer than the chamber-dust, and contained a number of lumps that were whitish in the interior. These had evidently been oxidized by slow combustion after they had been deposited. They resembled the burned "blue powder." Dried at 100° C., this sample gave:

Flue-Dust, No. 55,165.

	Per cent.
SiO_2,	2.54
PbS,	10.41
ZnS,	0.11
$PbSO_4$,	61.39
PbO,	11.47
ZnO,	0.42
Fe_2O_3,	0.97
Al_2O_3,	0.05
CaO,	5.24
MgO,	0.03
CO_2,	1.35
SO_2,	none
	93.98

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On combustion it yielded:

	Per cent.
CO_2	7.38
H_2O,	3.53

The volatilized compounds are too fine to settle in the dust-chamber to any extent. Drawn by a strong current of air, they pass out of the top of the chamber, and through a horizontal iron flue, 5 feet in diameter, carried 20 feet above the ground on iron pillars, to the fan in the engine-room, as shown in the plan. The current passes through the fan, and is forced through a 4-foot iron flue, supported every 12 feet upon iron pillars, to the first bag-house, which is called the "blue room."

This is a brick building, 95 feet long, 50 feet wide, and 45 feet high, and divided into two rooms by a longitudinal wall. Each compartment is separated into an upper and lower story by a floor 12 feet above the ground. Tie-rods in four rows, ten in each row, extend across the building from side to side. There are iron pipes, with bearing-plates at the ends, and the first row, 12 feet above the ground, serves to support the first floor.

The lower rooms have each ten windows, 6 by 2 feet, with ordinary sashes; while the upper rooms have the same number of windows, provided with iron shutters arranged to keep out rain while allowing the gases to escape. There is also a row of flues or openings along the center of the roof. The ground-floor in each room is formed by the flat tops of two rows of sheet-iron hoppers. Each row has ten hoppers and forms a flue. The hoppers serve to collect the accumulated fume. They extend to within $3\frac{1}{2}$ feet of the ground, where each is supported on four legs, and has a draw-plate to let out its contents. These flues, with hopper-bottoms, are of $\frac{1}{16}$-inch sheet-iron. There are sixteen holes above each hopper, and each hole has a 12-inch length of pipe 18 inches in diameter, upon which the lower end of a bag is slipped and tied. The bags are 33 feet long and 60 inches in circumference when new, but after hanging awhile they become 58 inches in circumference and about 35 feet long.

The tops are closed by tying with stout cords, by which they are also suspended from the frame-work above. There are 800 bags in the house, and they are quite expensive, having cost $9 apiece.

The bags are made of unwashed wool, this having been the mate-

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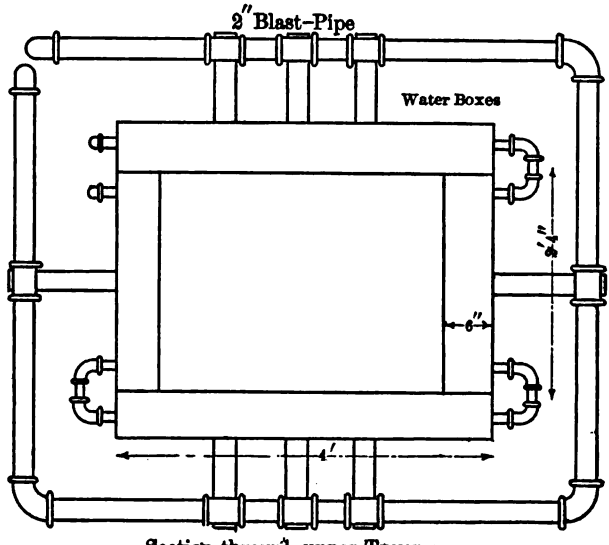
rial selected after a number of experiments. The acid in the fumes soon destroyed cotton, but it was found that wool, with the natural grease of the sheep still in it, was not so much affected.

The four hopper-flues extend along the building and connect with the main gas-flue, which delivers the fume and gases from the fan. The current passes up into the hanging bags, which are distended by the pressure, and the gases pass out through the cloth, leaving the fume upon the inner surfaces. As the fume accumulates, it drops into the hoppers, and is let out from time to time. The separation is apparently very perfect, and the gases in the upper chamber are clear and colorless.

FIG. 6.



Section through upper Tuyeres
SLAG-EYE FURNACE

The excessive amount of sulphur from the coal does not seem to affect the iron-work very much, and the bags are quite durable.

The lower chamber is accessible at all times for the purpose of emptying the hoppers. By having two divisions of the house one may be shut off from the current to gain access to the bags.

The bag-house requires but little attention, and merely an occasional visit. Once in about ten days the bags are shaken, and the accumulated fume is removed from the hoppers.

The first fume is called "blue powder." It is a very fine bluish-

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gray material, soft and unctuous to the touch, and very complex in composition. Its analysis presented some difficulties, which will be detailed further on. It contains sulphur as sulphides, sulphites and sulphates, besides carbonic and silicic acids, uncombined oxides and also hydrocarbons. Four samples, dried at 100° C. for two hours, gave:

Blue Powder.

	No. 55,166. 1st Hopper, 1st Row.	No. 55,167. 6th Hopper, 2d Row.	No. 55,168. 1st Hopper, 4th Row.	No. 55,169. 10th Hopper, 4th Row.
PbS	6.18	10.37	5.19	8.61
$PbSO_4$	45.34	43.55	46.88	43.57
PbO	44.44	44.48	45.08	44.18
Zn	.61	.34	.43	.61
$(FeAl)_2O_3$	.10	.05	.07	.07
CaO	.21	.01	.03	.02
SiO	.17	.11	.12	.14
CO_2	.23	.19	.26	.08
SO_2	.96	.12	.68	.44
On combustion yielded:				
H_2O	2.12	1.57	2.33	1.71
CO_2	3.62	2.98	3.22	2.92

After its removal from the hoppers, the fume or "blue powder," is piled in heaps upon the ground, and these are ignited by a few pieces of burning oil-waste. The burning takes about ten hours, and the crusts are then taken to the slag-eye.

The difficulty of handling this impalpable material would, perhaps, prove an effectual bar to its purification in the slag-eye, were it not for its inflammable nature. The piles of powder on the floor of the bag-house, after smouldering for a few hours, are converted into pinkish-white coherent crusts, free from all carbonaceous matter, and sulphide of lead. It is now in excellent condition for handling and charging.

The roasted or burned blue powder consists of oxidized compounds of lead. A sample, dried at 100° C., yielded:

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Roasted Blue Powder, No. 55,170.

	Per cent.
$PbSO_4$,	48.76
PbO,	46.82
Fe_2O_3,	0.32
Al_2O_3,	0.05
ZnO,	0.27
CaO,	0.48
SiO_2,	0.10
CO_2,	0.90
SO_2,	1.65
H_2O,	0.37
	99.72

The modified slag-eye, in which the fume is purified by a second volatilization, is 2 feet square and 6 feet high, and stands on the ground. There is a charging-opening, 4 feet above the ground. Eleven tuyeres are used, seven of which are just below the charging-opening, and four are at the bottom. The tuyeres are 1½ inches in diameter, and pass through water-boxes. They are straight and so arranged that a bar can be inserted to clear the opening. The construction of the furnace is shown in Figs. 4, 5 and 6.

There are three of these furnaces, arranged as shown in Fig. 1, two being operated at the same time. They stand in a row, beneath a flue which is 7 feet above the ground and 7 feet high by 4½ feet wide.

The furnace is kept filled up to the charging-door with a glowing bed of coke, upon which the materials to be smelted are thrown. The charge consists mainly of by-products of the works, but a small amount of oxidized ore is generally added. The by-products include the burned blue powder, dust and slag from the hearth-smelting, and fume from the cooling-pipes of the slag-eyes which is not suitable for paint. An average daily charge for one furnace is 2800 pounds of slag, 1000 pounds of blue powder, about 600 pounds of "dry-bone" and 450 pounds of fume from the cooling-flues.

Connellsville coke is used as fuel. The furnace is kept very hot, the flue above being highly heated. A hot flue seems to be necessary for the production of a good color in the paint. The product is inferior in quality for about twenty-four hours after starting the furnaces, or until the flue becomes hot.

The object in working this furnace is to volatilize as much as pos-

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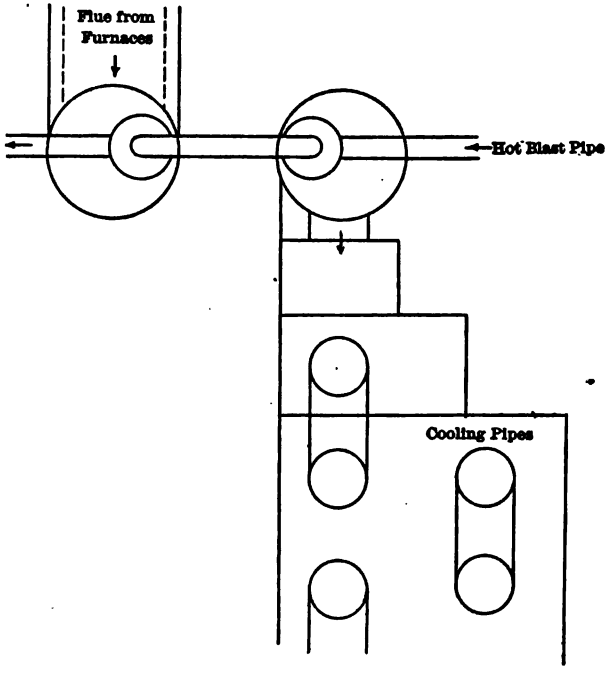
sible of the lead; but a certain portion is necessarily reduced, and this, with the poor slag formed, finds its way out through a hole in the bottom. Here is an iron basin having two compartments. The slag flows over some charcoal in one side and into a water-basin below. The lead filters through the charcoal and passes into the adjoining basin, from which it is ladled into moulds. Two furnaces consume, for this purpose, about five bushels of charcoal a day.

The lead-compounds are very corrosive and wear away the fur-

FIG. 7.



Plan of Cooling Cylinders etc.

nace rapidly; but, as differences in shape do not affect the product, the furnaces are run along for fifteen or twenty days, without stopping for repairs.

There are two men for each 8-hour shift, one feeder, and one man who ladles lead and carries away the slag. The latter uses a ladle and three bars about 6 feet long, while the feeder has two shovels and two short bars; one for cleaning tuyeres. He feeds the fire every four or five minutes, throwing in a light layer of coke, upon this the slag, and then the "blue powder."

The highly-heated gases from the slag-eye furnaces are drawn by

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an exhaust-fan through a series of iron cooling-pipes, and when the temperature is so far reduced as to avoid injury to the woollen bags, the gases are delivered to a second bag-house.

An average, per day, of thirteen pigs, or 1100 pounds, of lead is produced, which is quite impure and dirty, and is, therefore, re-melted in a casting-pot. This pot is similar to the one previously described. The lead is poled with wood to purify it before casting.

The slags are poor and are thrown away; but a small quantity is used over again in the furnace, being thrown against corroded spots in the walls, as a partial protection from the oxide of lead.

The final slag from the slag-eye is very dark-green, almost black, and glassy. While it is generally considered poor enough to throw away, yet it is very heavy and the analysis shows it to contain a considerable percentage of lead.

Final Slag, No. 55,179.

	Per cent.
SiO_2,	42.10
PbO,	25.37
FeO,	7.91
Al_2O_3,	9.58
Ni and Co,	traces
MnO,	0.27
ZnO,	4.48
CaO,	7.97
MgO,	1.66
S,	0.22
	99.56

An interesting solid deposit from the slag-eye flue showed:

Solid Flue-Deposit, No. 55,182.

	Residue. Per cent.	Per cent.
SiO,	2.57	
$PbSO_4$,	1.05	
Fe_2O_3,	.80	
Al_2O_3,	.63	
ZnO,	1.19	6.24
$PbSO_4$,		74.17
PbO,		8.52
Fe_2O_3,		.40
Al_2O_3,		.65
ZnO,		2.66
CaO,		4.45
MgO,		.22
Soluble SiO_2,		1.55
SO_3		.27
		99.13

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From the overhead flue of the slag-eye the gases first pass into an upright iron cylinder, 7 feet in diameter and 20 feet high, lined with brick. The 18-inch blast-pipe, which supplies the slag-eyes, passes through this, and also through a second similar cylinder, by which means the air is heated to a temperature that the hand will just bear. Tho two cylinders are shown in Figs. 7 and 8.

The current is drawn on through four inverted U-pipes, made of iron 3½ feet in diameter and 20 feet high. Their lower ends are set in brick-work, so that the gases ascend and descend four times through the exposed pipes, until they are quite cool. The arrangement is shown in Fig. 9. The gases then pass through the fan and on to the bag-room.

Some material settles in these cooling-pipes. It is not of good color, and not fine enough for paint, hence it is returned to the slag-eye.

Dust from First Cooling-Pipe, No. 55,183.

	Per cent.
$PbSO_4$,	89.84
PbO,	1.20
Fe_2O_3,	1.94
Al_2O_3,	0.32
ZnO,	2.68
CaO,	.27
Residue,	2.26
CO_2,	.13
SO_2,	.18
	98.82

This second bag-room, called the "paint-house" (Fig. 10), is also of brick, 40 feet wide, 90 feet long, and 45 feet high. It has three rows of hoppers, nine in a row, with twenty bags above each hopper. As the bags hang, each has 168 square feet of surface, so that the 528 bags present a filter-surface of 88,704 square feet.

There is no vertical division of the house. It is simply divided into an upper and a lower chamber. The lower floor is boarded, and has two wooden bins beneath the hoppers to receive the paint. The upper floor, 9 feet above, is supported on tie-rods of 2½-inch iron pipe, which extend between the side-walls, 9 feet apart. Eighteen upright posts of 3½-inch pipe, set in two rows down the house, support the weight of the floor; 1½-inch pipe is laid across the large pipes at intervals of 2 feet.

The three flues with hopper-bottoms are hung from the small pipes by sheet-iron straps, 12 inches apart, and walks are laid down

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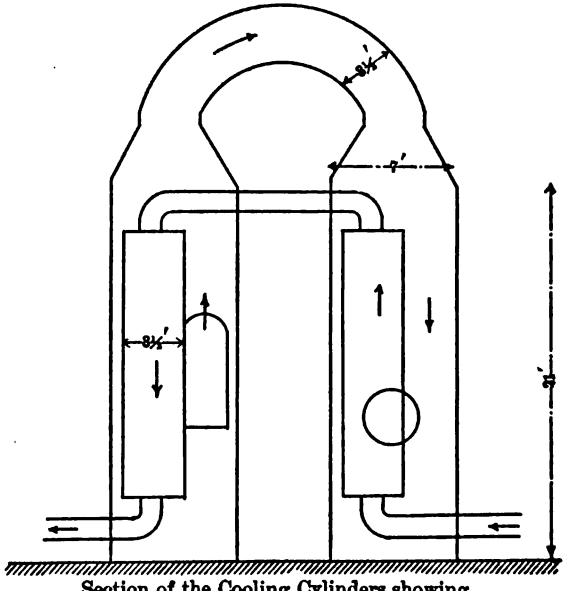
between the bags with plates of sheet-iron. The bags are hung from rods above. The roof is supported by four iron trusses, and has 21 escape flues, 18 inches in diameter, along the ridge.

The free use of pipe in the construction of the bag-houses is quite noticeable and has been found very satisfactory.

At the end of the building the main gas-pipe enters a distributing box, 10 feet wide and 30 feet long, with a hopper-shaped bottom. This opens into the three flues from which the gases pass to the bags where the fume is filtered out.

FIG. 8.



Section of the Cooling Cylinders showing the heating of the blast for the Slag-Eye Furnace.

The fume is in the form of a pure white, impalpable powder. It is shovelled out into a convenient box mounted on wheels, taken to the scales at the end of the room, and packed, 500 pounds in a barrel. Two men, provided with breathing-sponges, do this work.

There is little difference in the quality of the powder from different parts of the room, and it is all packed together, although that from the last bin of the third row is finer than that from the first bin of the first row, since the larger particles have been deposited first. Two samples, dried at 100° C., yielded:

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White Paint.

	No. 55,187. Per cent.	No. 55.189. Per cent.
Insoluble,	0.08	0.08
$PbSO_4$,	65.46	65.00
PbO,	25.85	25.89
ZnO,	5.95	6.02
Fe_2O_3,	0.03	0.03
CaO,	0.02	0.02
CO_2,	1.53	2.00
SO_2,	.04	none.
H_2O,	0.69	0.85
	99.65	99.89

The average product when two furnaces are running is 17 barrels, or 8500 pounds a day. The furnaces run for 15 to 30 days, and then stop for repairs, and for the removal of accumulations of slag and powder. In starting up with cold flues, the paint is so discolored that the current is turned off from 4 to 5 hours into the pipe leading to the blue-room. The material collected in the paint-room for the first 20 to 24 hours is kept separate, and classed as "A_o," being of inferior quality.

The two fans are side by side, and connection is made between the two pipes so that the current may be turned from one to the other.

The fans are of iron, 6 feet in diameter, 3 feet wide, and run at 290 revolutions a minute.

A No. 5 Baker blower supplies air through an 18-inch pipe to the slag-eyes; and another of the same size furnishes air to the hearth-furnace. Its capacity is far in excess of what is required for the hearth, so that most of the air is allowed to escape through a spring-valve.

The fans and blowers are run by a horizontal engine with 14-inch cylinder and 25-inch stroke. The cut-off is $\frac{1}{2}$, the steam-pressure 70 pounds, and the speed 84 revolutions a minute. The fans and blowers are some distance away, and the power is communicated by several lines of intermediate shafting. There is a Cameron boiler feed-pump, and an additional plunger feed-pump. The water-tank is 12 feet by 10 feet in size, and is elevated 25 feet on a stone platform. It is supplied by a No. 9 Blake pump. There are two boilers, of which only one is used at a time. Each is 28 feet long and 4 feet in diameter, with four 12-inch flues, and consumes about $2\frac{1}{2}$ tons of coal a day.

The chimney is of brick, 6 feet in internal diameter, and 95 feet high. One engineer for each shift attends to all the machinery.

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The daily cost of running the works is as follows:

	Each.	Total.
12 smelters at the hearth-furnace,	$2.00	$24.00
5 extra men " "	1.50	7.50
2 paint-packers, 25 cents per barrel,	4.25	8.50
3 engineers,	1.75	5.25
1 teamster,	1.50	1.50
2 foremen,	2.50	5.00
12 smelters at the slag-eye,	2.00	24.00
3 yard hands,	1.50	4.50
5 extra men,	1.50	7.50
		$87.75

The cost of the material when the collection was taken in 1884, was, for 48,000 pounds of mineral:

Lime, 48 pecks at 5 cents,	$2.40
Coal, 48 bushels at 7½ cents,	3.60
Coke, 2880 pounds at $8.15 per ton,	11.75
Charcoal, 10 bushels at 10 cents,	1.00
Engine coal, 2½ tons at $1.87,	4.75
	$23.50

The total daily cost may be thus estimated:

Labor,	$87.75
Materials,	23.50
General expenses, say,	13.75
Total cost of treatment,	$125.00

An experimental run was made upon 57,000 pounds of mineral, in which all the products were secured and weighed. The mineral was smelted in about one day and five hours, with the following results, 57,324 pounds mineral smelted, producing:

	Pounds.
Pig,	28,549
Slag,	11,140
Dust,	5,000
Blue powder,	13,690

All the slags and most of the blue powder were then put through the slag-eye furnaces.

Material charged.	Pounds.
Slag,	11,140
Blue powder,	13,226

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Products.	Pounds.
Pigs,	8,798
Poor slag,	4,030
Paint,	14,337
The cost of the mineral was,	$1169.38
Freight on the white lead to the East, at 45 cts. per 100 pounds,	64.51
Freight on the lead to St. Louis, at 24 cts.,	89.63
Commission on lead, 1 per cent.,	12.51
Royalty for the bag-patents, ½ ct. per pound,	71.68
Commission on white lead, 2.5 per cent.,	12.53
Cost of treatment,	158.87
Total expenditures,	$1579.11
Pig-lead sold at $3.35 per 100,	$1251.12
White lead sold at $3.50 per 100,	501.79
	$1752.91
Expenditure,	1579.11
Profit,	$ 173.80

To which must be added the value of 464 pounds of blue powder and 5000 pounds of dust.

The mineral was supposed to contain 68 per cent. of lead.

A description of a trial of the process in silver-lead smelting at the Grant smelter, Leadville, Colo., will be found in S. F. Emmons's *Geology and Mining Industry of Leadville, Colorado*, p. 673. An analysis of the material collected is given on page 717 of that work.

This process has been mentioned and more or less described as follows:

Mineral Resources of the United States, 1883–4, p. 427.

Engineering, 1884, p. 495. (Operations upon an argentiferous ore containing 8 per cent. lead and 30 per cent. of zinc, at Gunnison, Colo.)

Engineering and Mining Journal, vol. xl., p. 4, July, 1885. (A very good general description of the works by Mr. F. L. Clerc.)

B. und H. Zeitung, vol. xlvii., p. 346. (Describes the works of the Bristol Sublimed Lead Company, Bristol, England, where the process is in operation.)

Preuss. Zeitsch., vol. xviii., p. 195.

Fresenius Zeitsch., vol. viii., p. 148.

A qualitative examination of the various dusts and fumes showed them to be very complex and interesting chemically. Some of them

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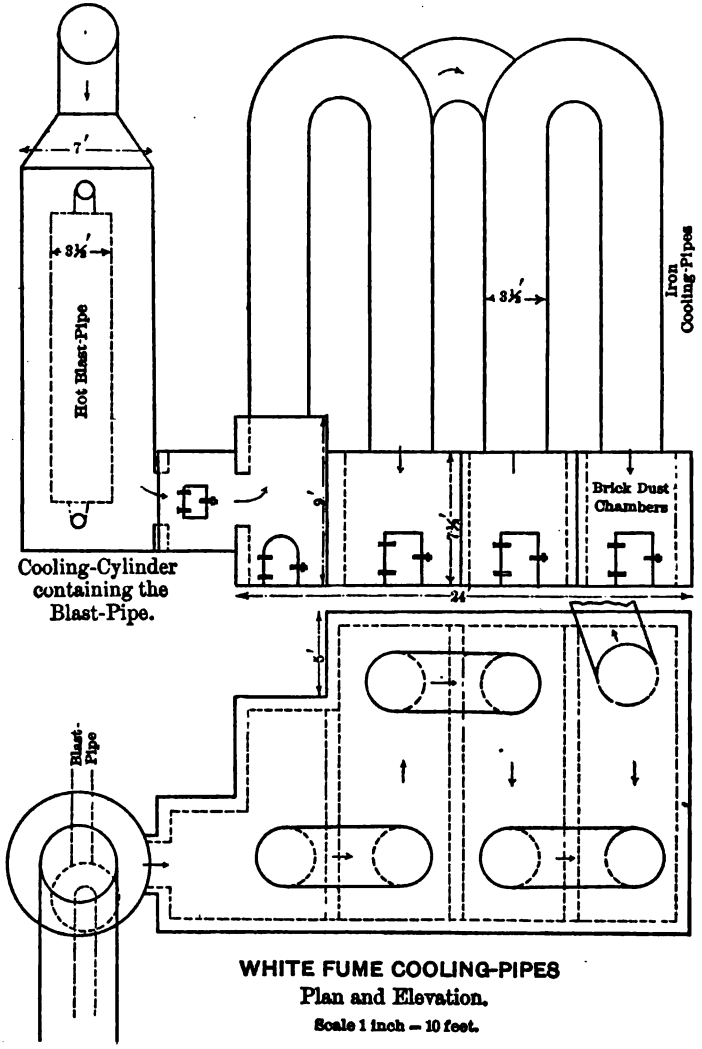
contain as acids, sulphur, sulphurous, sulphuric, carbonic and silicic acids; as bases, lead and zinc principally, with small amounts of iron, aluminum, calcium, and magnesium and traces of cobalt and nickel. Uncombined oxides are present in nearly every case.

FIG. 9.



WHITE FUME COOLING-PIPES
Plan and Elevation.
Scale 1 inch = 10 feet.

Owing to this complex nature, it seemed desirable to attempt, at least, to determine the amounts of the different compounds present. Much time was spent upon this; and while the results are not satisfactory in every case, yet they fairly indicate the distribution of the

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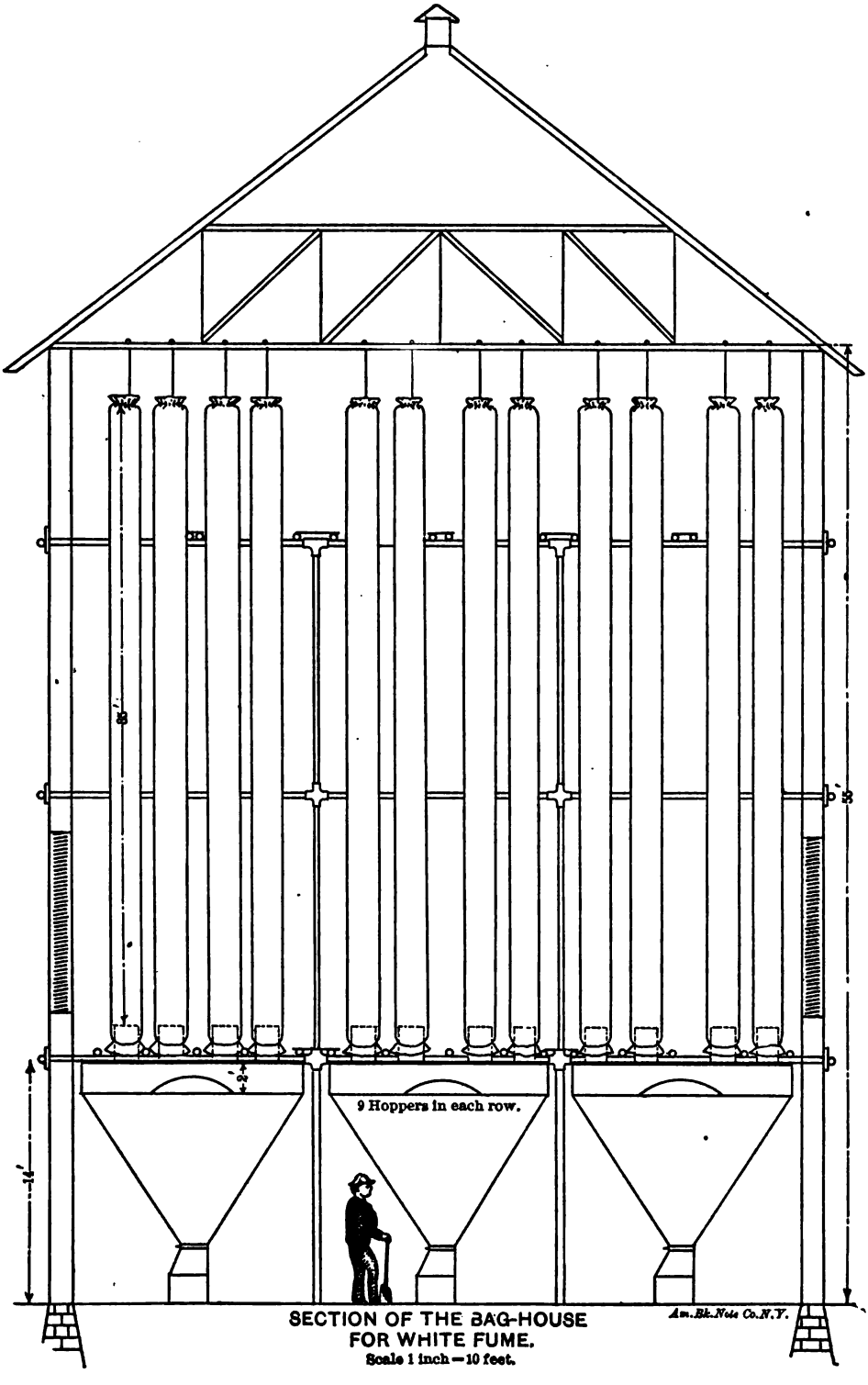
FIG. 10.



SECTION OF THE BAG-HOUSE
FOR WHITE FUME.
Scale 1 inch = 10 feet.

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acids and bases, and the figures given will enable any one to recalculate in any way desired. After much experimenting, the following general plan was adopted for the analyses of the fumes, dusts and rich slag.

About 1 gramme of the material was first treated with 1.5 to 2 grammes of bicarbonate of soda, to decompose the sulphate of lead. It is very difficult to make the solution of $HNaCO_3$ moisten the blue powders on account of bituminous matters they contain. Without being moistened of course the $PbSO_4$ can not be decomposed. By first treating the blue powders with a few drops of very dilute acetic acid this will moisten them and then the solution of $HNaCo_3$ can be readily applied. The small amount of acetic acid is neutralized and does not affect the action of $HNaCO_3$. After standing for several hours, the residue was filtered off and called A. The filtrate was made acid with HCl, boiled to expel CO_2, and the SO_3 was precipitated by $BaCl_2$. The excess of Ba was removed from the filtrate from the $BaSO_4$, and the solution then examined for bases. Generally no Pb was found; occasionally traces were indicated; only traces of Fe were found and generally only traces of Ca; but in a few cases a considerable amount was found.

"A" was treated with 10 c.c. of acetic acid of 1.04 sp. gr. diluted with 30 c.c. H_2O. The residue from this treatment was called B. This filtrate generally contained all the metals present in the material in an oxidized state, and, in some cases, SiO_2. They were separated and determined by ordinary methods. "B" was treated with 25 c.c. of dilute HNO_3 (1 HNO_3 sp. gr. 1.40 to 4 H_2O) which removed the metals present as sulphides.

The residue from this treatment was called insoluble residue. The metals in this filtrate were separated by ordinary methods.

In the absence of sulphides, as in the case of the white fumes and several of the dusts, the treatment with HNO_3 was omitted.

In a few cases, especially when the material had been subjected to a high heat, it was necessary to treat several times with $HNaCO_3$ in order to decompose the $PbSO_4$ entirely. In such cases, if the residue was considerable, it was fully analyzed.

In the absence of any considerable amount of sulphite, this method has yielded very satisfactory results. Since no considerable amount of SO_2 was found in any of the samples, the influence of SO_2 upon the method was not investigated.

The total sulphur was determined by treatment with Cl in alka-

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line solution. The difference between this and the sulphur found as sulphate and sulphide was called SO_2.

All the deposits from the hearth-smelting contain carbonaceous matter, the determination of which presented much difficulty; and finally all that was attempted was to submit the material to an organic combustion, and weigh the CO_2 and H_2O given off. Even this, however, was rendered unsatisfactory by the presence of sulphides, which might evolve SO_2 during the combustion. In the case of the blue powder, the excess of PbO present ought to hold any SO_2 formed; but with the dust this was not the case; and some measure had to be taken to prevent the SO_2 from getting into the potash bulbs, and being weighed as CO_2.

After some investigation, the use of PbO_2 was adopted. Carius* had proposed the use of PbO_2 to absorb SO_2 and used it for this purpose in an absorption-tube outside the combustion-tube. Warren† placed it within the combustion-tube, with good results. Kopfer‡ developed its application, and laid down a number of precautions necessary in its employment.

The method was found quite satisfactory, when carefully carried out in the absence of large amounts of SO_2; but the utmost attention must be given to trivial details, to insure success. In the case of the flue-dust, No. 55,164, containing a large amount of sulphides and some SO_2, it is an open question whether the PbO_2 completely absorbed the SO_2.

The chief difficulties arise from the attraction which the PbO_2 has for moisture, the power with which it holds on to it when once absorbed, and the ease with which it loses a part of its O on being heated, even gently, and becomes capable of absorbing CO_2.

If the PbO_2 is maintained at the proper temperature (150 to 180° C.), to prevent it absorbing H_2O during the combustion, then it will lose O and absorb CO_2. On the other hand, if it remains cool enough to retain all of its O, then it absorbs H_2O.

In order, therefore, to employ PbO_2 for this purpose, it is necessary that it be saturated with CO_2 and dry before starting the analysis, that it be kept cool during the combustion; that the H_2O absorbed be driven off by heating to 150° to 180° C., after the close of the combustion, and that the PbO formed during this heating be saturated with CO_2 before a new combustion is started.

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* *Ann. Chem. Pharm.*, 1860, vol. cxvi., p. 28.

† *Am. Jour. Sci.*, second series, vol. xli., 1866, p. 40. Paper before American Academy, March, 1865.

‡ *Fres. Zeit.*, vol. xvii., 1.

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In practice I prepared a combustion-tube 26 inches long as follows: The rear end for 6 to 8 inches was left blank for a long boat to hold the substance; next was a small asbestos plug; then 8 to 10 inches was filled with granular oxide of copper; next an asbestos plug 3 inches long, then 4 inches of granular PbO_2, and last an asbestos plug. Both ends were provided with tightly-fitting gum stoppers.

The tube having been prepared, a slow current of pure dry CO_2 is passed through it for one hour, the portion containing the PbO_2 being heated by an air-bath to 170° to 180° C. This will dry the PbO_2, and any PbO formed will be transformed immediately into carbonate. The tube is then cooled and filled with dry air. It is now ready for an analysis, which should begin at once.

The substance is weighed into a porcelain boat, thoroughly mixed with some pulverized dry CuO, and placed in the tube with the usual precautions. The tube is then filled with O and the combustion carried on in the ordinary way.

During the combustion the PbO_2 must be kept as cool as possible. Condensed water should be plainly visible upon the long asbestos plug between the PbO_2 and CuO. After the completion of the combustion, dry air is driven through the tube until nearly all the CO_2 has been removed. At the same time the heat is gradually reduced in the furnace. Next, the asbestos plug is carefully heated, to drive out the condensed H_2O, and at the same time the air-bath enclosing the PbO_2 portion of the tube is brought to 170° to 180 C., and kept there about half an hour.

The operation is now finished, and the potash-bulbs and $CaCl_2$ tubes are removed for weighing.

If all the details have been carefully attended to, we have the PbO_2 now dry as it was at the beginning of the combustion; it has not given off any CO_2 neither has it absorbed CO_2. The final drying has produced some PbO, which must be saturated with CO_2 before beginning a new combustion. A stream of dry CO_2 is passed through the tube, the CO_2 replaced by air and the tube is then ready for a new combustion.

The total sulphur was determined in the final slag by the Fahlberg-Iles* method of fusion in NaOH in a silver crucible.

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* *Fres. Zeit.*, vol. xvii., p. 497.

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PHOSPHORUS IN PIG-IRON, STEEL AND IRON-ORE.

BY CLEMENS JONES, HOKENDAUQUA, PA.

(Washington Meeting, February, 1890.)

THE analytical history of phosphorus in its relation to the metallurgy of iron is an interesting study, the progress of which runs parallel with the development of the greatest industry in the world. Without the means of finding out how much phosphorus crude pig-iron contained, the Bessemer steel process would have been impossible. To do this was originally tedious and slow work. On the other hand, the "direct" process of to-day, as well as the various special processes for steel or iron, have demanded the utmost rapidity in the operations of analytical chemistry.

In general, any rapid process for the conversion of unrefined metal into a product of standard composition depends upon the rapidity of some accurate means employed to control it. Moreover, in our modern practice, it is not improbable that the differences in physical properties among the various makes of iron or steel are affected, if not produced, by the use of different methods of analysis.

Recourse is always had for correction to the "standard methods," but I incline to think that these methods are susceptible to error. It is very certain that the system of duplicates would not be resorted to, if the fact were otherwise. Even when duplicated, under the most careful and skilful manipulation, these standard methods produce disagreements. In proof of statements like these, which may sound heretical, I hope, further on, to cite some evidence worthy of attention. I do not now mean to lay claim to the distinction of discovering the ultimate corrective. That result will come, no doubt, in time, as the product of the experience of many workers.

In the case of phosphorus, as of the other ingredients of iron or steel, a great deal has been done to accommodate the methods of analysis in vogue to the growing necessity for more rapid procedures. Much of this work has concerned what might be called the incidents of method. There is no better example of the accumulation of these incidental improvements than the work done on phosphorus; and I

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believe that the final successful outcome of such progressive changes will be properly considered the standard method.

The ideal analysis, both for refined conditions and delicate determinations, is to be found, in nearly all instances, in a volumetric method. But, in the preparatory stages, the treatment by this standard method must consist of *visible reactions.*

I have classified what appear to be the salient features already embodied separately in the different methods; and when so related, they apparently fulfil the conditions and the object that each has had in view.

The method of Mr. F. A. Emmerton (*Trans.*, xv., 93), is the real beginning of this progressive series. As recommended, certain objections were found to it. The first of these was in the use of dilute nitric acid as a wash for the yellow precipitate. Acid ammonic nitrate, as suggested by Dr. Drown, was then substituted, with only partial success. The presence of nitric acid in the final solution, reduced for titration, seriously interfered with accurate work. I communicated this fact to Mr. A. L. Colby, chemist to the Bethlehem Iron Co., who at the time was also experimenting. His conclusion, reached by careful examination, pointed out the remedy. I quote from Mr. Colby's notes, kindly placed at my disposal:

NOTES ON EMMERTON'S METHOD FOR PHOSPHORUS, OCTOBER, 1887.

Blank Phosphorus Determination.—A clean dry 16-oz. Elrymeyer flask was washed twice with am. nitrate wash (NH_4NO_3, 200 gms.; HNO_3 conc., 30 cc.; H_2O, 2000 cc.), pouring it on a filter, and draining the flask as much as possible each time; the paper was then washed twice with the wash, and sucked dry; the stem of the funnel was rinsed off on the outside with water, and the funnel placed in the flask; paper punctured with platinum rod, and washed with dilute NH_4OH(1 : 4), about 30 cc. being used; 5 gms. granulated zinc was added to the contents of the flask, and 80 to 90 cc. H_2SO_4 of the dilution, 150 H_2SO_4 : 420 H_2O). A small funnel was put in the top of the flask, which was then heated till nearly all the zinc was dissolved. The reduced solution was then filtered, flask, funnel and paper were thoroughly washed with hot water, and the solution was titrated with $KMnO_4$ at once.

A permanent tint was not obtained until 12 cc. of $KMnO_4$ had been used. (As 1 cc. of $KMnO_4$ = .000106 gms. P, this is equivalent to an error, in analyzing an ore where 8 gms. are taken, of 0.016 per cent., and for a steel or iron where 4 gms. are taken, of 0.032 per cent., P).

Above blank test, using as a wash HNO_3 (1 : 25). Permanent tint at 8.5 cc. of $KMnO_4$.

Above blank test, using as a wash HNO_3 (1 : 50, which Emmerton recommends). Permanent tint at 3.2. cc. of $KMnO_4$.

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Above blank test, using as a wash H_2SO_4 (1:50). Permanent tint at 0.45 c.c. $KMnO_4$.

Above blank test, using as a wash $(NH_4)_2SO_4$, made up as follows: $(NH_4)_2SO_4$, crystals, 25 gms.; H_2SO_4, conc., 50 c.c.; H_2O, 2500 c.c.
Permanent tint at 0.45 c.c. of $KMnO_4$.

Zinc 5 gms. and H_2SO_4 (90 c.c.), placed directly in a dry clean flask, reduced as usual, filtered, washed and titrated. Permanent tint at 0.45 c.c. of $KMnO_4$.

To ascertain if yellow precipitate is soluble in H_2SO_4, or in $(NH_4)_2SO_4$ wash, which would prevent its use as a substitute for NH_4NO_3 wash. Selected crystals of Na_2HPO_4. $12H_2O$, containing by theory 8.6572 per cent. P, were dissolved, 1.5408 gms. in 500 c.c. distilled water, and the following tests were made:

Tests with Various Wash-Waters.

Date. 1887.	Test No.	Wash-water used on yellow precipitate.	Na_2HPO_4 solution taken c.c.	$KMnO_4$ used. c.c.	Correction for zinc.	Phosph'rus per cent.
Sep. 20	1	Ammon. nitrate.....	10	27.4	0.5	9.252
" 20	2	" "	10	26.7	0.5	9.012
" 20	3	Ammon. nitrate, and then washed twice with water to remove NH_4NO_3.....	10	25.8	0.5	8.702
" 21	4	H_2SO_4 (1:5).....	10	25.7 Zinc ran through.	0.5	8.668
" 21	5	" "	10	25.3	0.5	8.530
" 20	6	" (1:50).....	25	64.6	0.5	8.8195
" 20	7	" "	25	64.6	0.5	8.8195
" 21	8	" "	10	25.0	0.5	8.42
" 21	9	"	10	24.9	0.5	8.39
" 21	10	$(NH_4)_2SO_4$.....	10	25.7	0.5	8.668
" 21	11	" "	10	25.7	0.5	8.668

As the blanks run both with H_2SO_4 wash (1:50) and with $(NH_4)_2SO_4$ wash used up 0.45 c c. $KMnO_4$ instead of 0.5 c.c, as subtracted above, analyses Nos. 4, 10 and 11 should show 8.685 per cent. instead of 8.668 P.

The above analyses, using 10 c.c., were made on too small a quantity of Na_2HPO_4. Small variations in $KMnO_4$ make too large differences in per cent. of P, 0.5 c.c., or one drop of $KMnO_4$, being equal to .007 per cent. P.

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* But the set of analyses shows that even strong H_2SO_4 (1 : 5) exercises no appreciable solvent action on yellow precipitate, and that $(NH_4)_2SO_4$ wash is rather better in agreement than H_2SO_4 (1 : 50).

As an example of the possibilities of the method when using the $(NH_4)_2SO_4$ wash for determination of small amounts of P, a solution of C. P. Fe_2Cl_6 was tested. Of this solution, which had been prepared from the filtrate from P determinations in pig-iron, by precipitation with NH_4OH, thorough washing, and solution in HCl, 25 c.c., containing about 2.5 gms. Fe, were precipitated for P as usual; the cloud of yellow precipitate was filtered off, washed with H_2SO_4 (1 : 50), dissolved, reduced, etc., as usual, titration giving only 0.75 c.c. $KMnO_4$, which, after zinc correction of 0.5 c.c., equalled 0.001 per cent. phosphorus—a result not obtainable when using NH_4NO_3 wash.

In addition, the following tests of ores were made, using $(NH_4)_2SO_4$ wash for the volumetric determinations, and washing four times on the paper (instead of twice, to test the solvent action, if any) and twice in flask.

Sept. 23.—*Filtration by suction*, taking the usual precaution of draining flasks and funnels, and washing stem of funnel.

10 gms. of ore dissolved; $\frac{5.0}{100}$ taken for phosphorus.

		P, per cent.	
No. 1524, . . .	20.9 c.c. $KMnO_4$	= 0.027.	After deducting 0.5 c.c., correction for Zn.
" 1525, . . .	14.8 c.c. "	= 0.020.	
" 1526, . . .	7.6 c.c. "	= 0.010.	

Sept. 28.—*Gravimetric determination* on 10 gms. ore, $\frac{5}{100}$ taken for phosphorus, using NH_4NO_3 wash, twice in funnel, and twice in flask—other conditions as before, but making two precipitations of NH_4MgPO_4 before weighing as $Mg_2P_2O_7$.

	$Mg_2P_2O_7$. Gms.	P. per cent.
No. 1524, ...	0.0079	= 0.025
" 1525, ...	0.0062	= 0 022
" 1526, ...	0.0026	= 0.009

Mr. Colby's notes close with some additional tests which confirm the foregoing.

Concerning the use of acid ammonic sulphate, it is interesting to note that, nearly a year later, Mr. Porter W. Shimer, in a paper before the Institute (*Trans.*, xvii., 100), observes: "Sulphuric acid and *ammonium sulphate* completely take the place of nitric acid and ammonium nitrate."

In the same paper Mr. Shimer describes the use of potassium permanganate as an oxidizing agent. This appears to be the next important detail. By means of it the complete oxidation of the phosphorus to ortho-phosphoric acid, as Dr. Drown puts it, is accomplished. This is indispensable, and cannot be omitted in the solution of pig-iron, steel or iron-ore.

Dr. Drown's method (*Trans.*, xviii., 90) of dissolving pig-iron or

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steel in nitric acid of 1.135 specific gravity, in which potassium permanganate, as suggested by Mr. Shimer, is used as an oxidizing agent, is a most happy expedient. It opens the way to the most rapid method in use. There is no ground to doubt its efficacy, if applied to iron-ore, by taking advantage of a simple device, namely, solution in dilute sulphuric acid of the fused residue from the ore. According to Dr. Drown, silicon exerts no influence on the determination of phosphorus in pig-iron or steel. As a *soluble silicate* there is good evidence of its non-interference in the treatment of iron-ore. A determination in iron-ore can thus be made in nearly the same time as in iron or steel.

Here, then, seem to be collected nearly all of the requisite conditions. There is but one remaining, the reduction of the molybdic acid. I believe that this is fully covered by using the method discovered by me for reducing ferric solutions (*Trans.*, xvii., 411). I have found that it applies in all respects equally well to the complete and instantaneous reduction of molybdic acid.

It is possible then to reconstruct a method, based on these incidents, which should be peculiarly adapted to every demand for the rapid and accurate determination of phosphorus.

Before advancing so far, however, I propose to show with what degree of accuracy these principles may be employed, by giving the following results. These were obtained by the Emmerton method, modified by the use of ammonic sulphate in washing the yellow precipitate, and by the use of the reductor.

No. 3588.—*Pig-Iron.*

Weight. gm.	c.c. $KMnO_4$. c.c.	Correction. c.c.	Phosphorus. Per cent.
.5000	18.67	0.03	0.447
.5000	18.52	0.03	0.443

1 c.c. $KMnO_4 = 0.00012$ P.

No. 3980.

	Phosphorus. Per cent.
No. 1, gravimetric,	0.717
No. 2, volumetric,	0.720

The following comparison is between this method (called "Reductor") applied to iron-ore, and the Emmerton method of adding the zinc directly.

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No. 4260.

	Weight taken. gms.		$KMnO_4$. c.c.	Phosphorus. Per cent.
A,	1.9700	Emmerton,	16.9 —0.2	0.101
B,	2.6078	"	20. —0.2	0.091
C,	2.8800	Reductor,	22.15—0.05	0.092
D,	3.0760	"	23.81—0.05	0.092

No. 4261.

	Weight taken. Gms.		$KMnO_4$. c.c.	Phosphorus. Per cent.
A,	1.6901	Emmerton,	30.01—0.2	0.212
B,	1.9439	"	34.5 —0.2	0.211
C,	1.9915	Reductor,	36.9 —0.05	0.222
D,	1.9982	"	37.1 —0.05	0.222

In the series following, carried through like the foregoing, potassium permanganate was used to oxidize the solutions. This was done carefully, before adding the ammonia. The improvement is marked.

No. 3681.—*Pig-Iron.*

Weight. Gms.	$KMnO_4$. c.c.	Correction. c.c.	Phosphorus. Per cent.
.5025	2.83	0.05	0.066
.5030	2.72	0.05	0.063
Determination by Mr. Shimer, by his Mb-Mg method,			0.064

The small amount of the sample necessitated the low weights, which, in that measure, affected the determination.

No. 3680.—*Pig-Iron.*

Weight. Gms.	$KMnO_4$. c.c.	Correction. c.c.	Phosphorus. Per cent.
.5000	26.39	0.05	0.6321
.5000	26.4	0.05	0.6324
Determination by Mr. Shimer, by his Mb-Mg method,			0.615

No. 3682.—*Pig-Iron.*

Weight. Gms.	$KMnO_4$. c.c.	Correction. c.c.	Phosphorus. Per cent.
.2500	23.21	0.05	1.111
.2500	23.21	0.05	1.111
Determination by Mr. Shimer, by his "sulphuric-acid" method,			1.07

The conclusion to be drawn, is not that the direct use of zinc always gives uniformly low results (by reason of the higher correction), but that the same errors are introduced, as I have pointed out in the case of the reduction of ferric solutions.

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Mr. Emmerton remarks that the "port-wine" color is evidence of only incipient oxidation. On the contrary, this oxidation is very appreciable. It is only otherwise (*i.e.*, the molybdous acid oxidizes slowly), as shown by Wernecke, when completely reduced.

The color of these solutions, as ordinarily diluted, is always olive-green. By using the reductor, this color is invariably obtained. The following result, from a piece of iron, shows that it is applicable to very high-phosphorus iron:

Weight. Gms.	$KMnO_4$. c.c.	Correction. c.c.	Phosphorus. Per cent.
.5033	177.35	0.05	4.227

The color of this solution was almost opaque until diluted. It was reduced instantly.

Dr. Drown's method, I have found capable of being modified to great advantage. A large number of trials were made, in strict accordance with the method as described, but neither the organic salts nor the acids gave acceptable results. Suspecting the trouble to lie in the use of these, I substituted ferrous sulphate, which has the advantage of dissolving the manganic oxide immediately, and being simultaneously oxidized. But the drawback to its use was that a correction for phosphorus had to be applied. None of the samples of ferrous sulphate received from a number of sources as "strictly C. P." were even low in phosphorus.

Messrs. Baker and Adamson, of Easton, Pa., kindly offered to experiment, and have succeeded in making the salt free from phosphorus. This was used in all of the determinations following.

The temperature of 90° C. was also found to be too high, molybdic acid precipitating in consequence. By adhering to 85° C., as recommended by Mr. Emmerton, this was overcome. Certain irons, high in manganese or fine graphite, cannot be easily oxidized before filtration. It is advisable always to test the solution with a few drops of $KMnO_4$, before adding the molybdate solution.

The method for pig-iron and steel now stands as follows:

Dissolve in nitric acid of 1.135 specific gravity, according to Dr. Drown; allow the solution to boil one minute, then add the solution of potassium permanganate; after MnO_2 appears, add a few small crystals of ferrous sulphate. Filter solution into flask. Add sufficient NH_4OH (.90 specific gravity); when solution clears up, add a few drops of $KMnO_4$, to insure oxidation, again dissolving, if necessary. At 85° C. add to the solution 75 c.c. ammonic molyb-

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date solution (Dr. Drown's formula), and agitate violently for five minutes. Filter in Bunsen filter-flask, and wash thoroughly with ammonic sulphate, observing all the precautions indicated by Mr. Colby. Dissolve the yellow phospho-molybdate in ammonia (.96 specific gravity), *without* puncturing paper. Wash once or twice with water. Suck dry, and remove flask and stopper to suitable bell-glass. Pour the ammonia solution into small beaker, washing out flask with water three times. Place this flask under the bell-jar and again filter the ammonia solution into it, this time washing paper thoroughly with water. Wash the zinc in the reductor, as required for ferric solutions. Then add 30 c.c. to 50 c.c. sulphuric acid (1.32 specific gravity) to the ammoniacal solution, and filter through the reductor by strong suction, washing the same as indicated for ferric solutions, and titrate as usual. The correction for zinc is obtained as described. I have made determinations in this way in half an hour. The following results were obtained, in the above manner:

The first is a "standard steel" which has been repeatedly tried by various chemists. The phosphorus is considered to be between 0.064 and 0.065. Below is a table, giving the former results, and the methods employed, the chemists being indicated by letters:

No. 4768.—*Standard Steel.*

Chemist.	Phosphorus. Per cent.	Method.
A,	0.063	Emmerton.
"	0.064	"
"	0.065	"
B,	0.063	"
"	0.065	"
C,	0.063	Not given.
"	0.065	" "
"	0.065	" "
D,	0.0603	" "
E,	0.061	Weighs yellow ppt.
F,	0.0666	Magnesia method.
"	0.0650	" "
"	0.0651	Weighs yellow ppt. original.
G,	0.0660	" " " "
H,	0 061	Magnesia method.
"	0.065	Titration of solution from the same NH_4MgPO_4.

This steel yielded, by the method I have above described, the following results:

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Weight taken. Gms.	$KMnO_4$. c.c.	Correction. c.c.	Phosphorus. Per cent.
3.1264	16.89	0.05	0.0646
3.1267	16.86	0.05	0.0645

The following determination was made in fifty minutes from the start:

Weight taken. Gms.	$KMnO_4$. c.c.	Correction. c.c.	Phosphorus. Per cent.
3.5014	18.52	0.05	0.0633

Value of 1 c.c. $KMnO_4$, 0.00012 gms. phosphorus.

A sample of Swedish pig-iron was tried in like manner. The former determinations were:

No. 5009.

Method.	Phosphorus, Per cent.
Emmerton,	0.023
"	0.023
"	0.023
"	0.023
Magnesia Gravimetric,	0 023
" "	0.022

This gave the following by my method:

Weight, Gms,	KMnO, c.c.	Correction, c.c.	Phosphorus, Per cent.
3.5187	6.87	0.05	0.0232

A second determination, completed in fifty minutes, P = 0.024.

The method was also applied to iron-ore. This was dissolved in hydrochloric acid (1.12 sp. gr.) and filtered. The filtrate was evaporated with nitric acid in the usual way. The residue was fused with excess of sodium carbonate, and then dissolved in the original beaker in dilute sulphuric acid and water. This solution was added to the nitric acid solution, which was transferred to a flask, oxidized with potassium permanganate, and then treated like the iron solution.

No. 4995.—*Rubio Ore.*

Former Determination.	Phosphorus, Per cent.
Emmerton Method, residue not fused,	0.012
" " " fused,	0.011
I. By method described above,	0 013
II. In way described for results, Nos. 4260–61, this ore gave	0.0118

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Another Bessemer ore, of which the former determinations were as follows:

No. 5008.

	Phosphorus, Per cent.
Emmerton Method, not fused,	0.016
" "	0.016
" "	0.016
Gave the results by the methods referred to in No. 4995, I. and II.:	
I. Method,	0.017
II. "	0.018

There is no loss of time in fusing the residue by this plan, and it is a desirable precaution under any circumstance.

In pig-iron or steel, arsenic and titanium interfere with the accuracy of the rapid method. The former must always be separated, and retards the process materially. In fact, the presence of arsenic may be regarded as an obstacle to the determination of phosphorus by any method.

Titanium, probably as titanomolybdate, has an apparent influence on the determination of phosphorus by the rapid method. When the Emmerton method of evaporating the original solution to hard dryness is used, this influence is lessened, but if titanium be present in any appreciable amount, the results are vitiated. The following case by each method is a good example:

No. 4706.—*Pig-Iron* (*containing Titanium*).

	Phosphorus, Per cent.
By Mr. Shimer, Mb–Mg method,	0.659
By the Emmerton method, modified as in 3680–81–82,	0.649
By the " "	0.643
By the " "	0.654
By the " "	0.650
By the " "	0.647
By Rapid Method,	0.726
By " "	0.719

It may not be amiss to refer to the importance of using a freshly prepared solution of the molybdic acid, which should stand twenty-four hours, at least, before use. Dr. Drown's formula (*Trans.*, xviii., 94), has given the best result after several years' trial. The solution must be filtered through asbestos as required.

A summary of the procedure may be found convenient:

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Rapid Method for Pig-Iron or Steel.

I. Dissolve in 60 c.c. nitric acid (sp. gr. 1.135).
II. Boil one minute (after action ceases).
III. Add 5 c.c. potassium permanganate (or until MnO_2 precipitates).
IV. Dissolve the MnO_2 with ferrous sulphate* (a few grains usually).
V. Filter into flask (500 c.c. capacity).
VI. Add 10 c.c. NH_4OH (sp. gr. 0.90), or nearly neutralize.
VII. Test for oxidation (III. and IV.). This is done while solution is heating.
VIII. Add 75 c.c. molybdate solution at 85 °C., and shake vigorously for five minutes.
IX. Filter and wash with ammonic sulphate.
X. Dissolve in NH_4OH (sp. gr. 0.96). Observe precautions.
XI. Add 30 to 50 c.c. sulphuric acid (1 : 2 water) and filter through reductor, observing precautions mentioned (*Trans.*, xvii., 414).
XII. Titrate.

Rapid Method for Iron-Ores.

I. Dissolve in hydrochloric acid (1.12 sp. gr).
II. Filter. Evaporate filtrate with nitric acid (1.20 sp. gr).
III. Fuse residue with excess of sodium carbonate, and dissolve in dilute sulphuric acid (1 : 2 of water). Combine solutions when the filtrate with nitric acid is ready. Then proceed as above, from III. to end.

An accurate determination of phosphorus in pig-iron or steel, in cases uncomplicated by arsenic or titanium, can thus be made in less than an hour. For iron-ore but little more time will be necessary.

The three vital conditions of the volumetric method, solution, oxidation and reduction, by visible reactions, give the method great advantages over any other. From the number of trials made with it, the conclusion seems unavoidable, that, compared with the "magnesia" method, the results are quite as accurate, and certainly more closely accordant. The principles are correct, and I believe that the method can be demonstrated to possess all the requisites for filling the difficult role of a standard method.

I wish to acknowledge my indebtedness to Dr. Drown, Mr. A. L. Colby and Mr. Porter Shimer for samples and valuable information.

* This may be obtained of Baker & Adamson, Easton, Pa., free from phosphorus.

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THE UNITED STATES PROTOTYPE STANDARDS OF WEIGHT AND MEASURE.

BY PROF. T. C. MENDENHALL, SUPERINTENDENT OF THE UNITED STATES COAST AND GEODETIC SURVEY.

(Address at the Washington Meeting, February, 1890.)

ALL persons, actively engaged in your profession, must have a natural interest in the subject of weights and measures. All members of the engineering profession have to do with operations of weighing and measuring—some with measuring more than weighing, and some with weighing more than measuring—but all with both of these processes to a greater or less degree. Methods of weighing and measuring have improved constantly from the earliest times and have kept pace with the advance of other technical processes. It is, therefore, believed that you will be interested in a few remarks concerning some of the more recent advances in relation to the establishment of decimal standards of length and mass in this country. Before speaking of this, I will refer very briefly to some facts which may not be very generally known regarding our customary system of weights and measures.

Although the yard is universally accepted as the unit of length, and the pound as the unit of mass, it is not generally understood, I believe, just in what respect the yard is the unit of length, or the pound the unit of mass. That is to say, it is not generally known in what degree these units have been legalized by Acts of Congress. Doubtless, many of you are familiar with the facts which I will briefly recapitulate.

The Constitution of the United States authorizes Congress to establish a system of weights, measures and coinage. The establishment of a system of coinage, as you know, was one of the first acts of the government, and the system then adopted is certainly one of the most perfect, if not the most perfect, that the world has yet seen. But, although Congress had the power to establish a system of weights and measures, it has never yet seen fit to exercise this power; that is to say, Congress has never yet

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passed a law declaring that the yard shall be the standard of length throughout the United States and that the pound shall be the standard of mass. In what way, then, have these units acquired the recognized legal standing which they unquestionably possess? To this question I will reply, that it has come about almost exclusively through the action of the State governments, as encouraged and assisted by the National government. In the early part of the present century, the subject of weights and measures was involved in great confusion throughout the whole country; the States did not agree among themselves, nor were the same units or standards of weight and measure used exclusively in any one State.

It soon became evident, however, on account of the very important and extensive operations of the general government in the survey of its boundary-lines and coasts, and also in the collection of duties upon imported goods, that it was important for the government to select or establish a unit of length and a unit of mass, which might be used in these operations, and in others in which the government was engaged. The business of surveying the coasts and of collecting the duties was then, as it has been almost continuously, under the direction of the Secretary of the Treasury. The establishment of a unit of length was virtually entrusted to Mr. Hassler, the first Superintendent of the Coast Survey, who selected a certain number of inches—or, rather certain particular inches—from a scale known as the Troughton scale, which is now in the archives of the Coast and Geodetic Survey. Ten or fifteen years later (about 1828), what is known as the Troy pound, which was brought to this country under the direction of Albert Gallatin, during the Presidency of John Quincy Adams, was selected as the unit of mass for purposes of coinage. This was done by definite Act of Congress, and not by the mere direction of a department official, and in this respect this standard of mass takes a higher rank than does the standard of length. This Troy pound was adjusted to agree as closely as possible with the British standard, by the celebrated Kater, and was brought to this country sealed, and shortly after its arrival the seals were broken with considerable ceremony by President Adams. This standard is at present, and has been during most of its life, deposited in the Mint at Philadelphia. It is an imperfect standard in its construction and in its form, and is by no means up to the requirements of modern metrological science. But, in 1836, in order to encourage uniformity in weights and measures

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throughout the whole country, Congress passed an act providing for the construction, by the Bureau of Weights and Measures, which has always been under the direction of the Superintendent of the Coast and Geodetic Survey, of accurate copies of these standards, together with capacity-measures of various kinds, which were to be distributed to the various States in the Union. The effect of this distribution has been to insure practical uniformity in weights and measures throughout the country. In the absence of positive legislation by Congress upon the subject, nearly, if not quite, all of the States have enacted laws in which they have adopted the standard yard, or rather each its own copy of the Treasury standard, as the standard of length for the State, and a standard of mass which is, also, a copy of, or derived from, the Treasury standard.

From this bit of history, with which, doubtless, many of you are already familiar, you will learn that, while Congress has never really legislated to provide these very important and necessary standards for the use of the whole country, practically the same result has been attained through the adoption by the State legislatures and general assemblies of the distributed copies of the Treasury originals. It is to be regretted, however, that this leaves us in the condition of having, in this country, a multiplicity of standards—forty or fifty, perhaps,—instead of a single standard, which would have been theoretically, if not practically, more desirable. It is an interesting and important fact that the first standards of length and mass, which were really legalized by Act of Congress for the whole country, were the metric standards. In 1866, an act was passed, making the use of the metric system permissible in the United States, and establishing legal equivalents among the various units and denominations of that system and the customary system. By Act of Congress, it was also provided that each of the institutions of learning, known generally as the land-grant colleges, should be provided with a set of metric standards, including a meter and a kilogramme, with the multiples and sub-multiples of the latter, and also with the metric measures of capacity. As Congress had acted upon the matter, no State action was necessary to legalize the use of this system, and I believe that none was had by any of the States. The Bureau of Weights and Measures is in possession of many interesting and accurate copies of the meter of the archives, and from these the standards thus distributed, were obtained.

The next important step was taken in the year 1869, when the French Government issued an invitation to various countries to send

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delegates to an International Commission to meet in Paris in the summer of 1870. The object of this Commission was to consider the question of constructing an international standard meter which should represent with all possible accuracy the actual length of the meter of the archives, and which should be more in accordance with the demands of modern metrology. At the short session which was held in 1870, it was agreed by this Commission that a committee should be appointed to make experimental researches, especially with the view of determining the best form for the international prototypes and the best material for their construction. The United States was represented upon this committee, which began its operations early in the year 1872. In the autumn of that year the International Commission met again, and at that meeting twenty-five nations were represented by forty-three delegates. It was there determined that the international meter should have the length of the meter of the archives at zero Centigrade, and that it should be composed of an alloy of platinum and iridium, 90 per cent. being platinum and 10 per cent. iridium. It was also agreed that the composition of the kilogramme should be the same as that of the meter; that its mass should be derived from the kilogramme of the archives in its actual present condition, and that it should be determined with reference to its weight in a vacuum. It was also decided to enlarge the scope of the Commission by the establishment of an International Bureau of Weights and Measures, and this proposition having been submitted to the various governments, a treaty to that effect was signed at Paris in 1875. Previous to this time, the experiments of the committee, charged with construction, had led to the production of what is known as the alloy of 1874, which was cast in a single ingot weighing 250 kilogrammes.

Although upon analysis, this alloy was found to contain such impurities that it was thought best by the International Commission to reject it, yet certain standards or units were prepared from this alloy, and one of these is in the possession of the Office of Weights and Measures in Washington. It was not until about the year 1882 that the difficulty in the way of preparing raw material for the construction of these standards was overcome. The method of preparing this alloy is described as follows: For the kilogrammes pure platinum and iridium in a finely powdered state was weighed in the proportions sought, and well mixed in quantities of 10 kilogrammes at a time. Each of these quantities was compressed into a cake and heated to a red heat in a covered platinum crucible. Each cake

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was then put into a furnace of pure lime and melted with an oxy-hydrogen flame and then poured into moulds, also made of pure lime. After various other processes, the ingots were forged under a powerful hammer. After forging, the metal was passed between polished steel cylinders and reduced to plates about 2 millimeters thick. Finally, all the metal was melted, as before, for a third time in May, 1884, and cast in a single ingot weighing 65 kilogrammes.

The metal thus obtained, was heated in a specially constructed lime-furnace, then forged under a hammer into a square bar 52 centimeters long by 7 centimeters thick. This, in turn, was heated and rolled between steel cylinders into a cylindrical bar 200 centimeters long and 44 millimeters thick, and at this stage pieces were cut off the ends and analyzed, and found to be pure. This bar was cut into forty cylinders for the standard kilogrammes.

The metal for the meter-bars was prepared in about the same way. After the several meltings, the mass was divided into five lots. These having been examined and found to be pure and homogeneous, were each divided into three parts. These were then arranged in groups made up from each of the five lots, melted together and cast into ingots, each of which contained sufficient material for one standard. The bars, after having been forged, were passed between rollers until they assumed approximately the shape desired. This operation required 3 days, 16 hours each, for each bar; *i.e*, two bars a week were planed, saving accidents. The bars were then cut to exact length (1.02 meters), their edges were made straight and the neutral surface was made plane. The polishing and tracing was then done at the Conservatoire des Arts et Métiers.

Great care was taken in ascertaining the coefficient of expansion of all of the bars after their preparation, one being selected and studied by the usual method for determining this constant and the coefficient of the other bars being determined relatively to this one by comparison at various temperatures. Thirty bars were prepared and very carefully compared with each other, and it was found that none of them differed as much as three one-thousandths of a millimeter from the meter of the archives. That one which was closest in agreement and which was altogether the best for the purpose, was selected by the committee and set aside as the international standard meter, thus taking the place, as a matter of fact, of the meter of the archives. The remaining bars were distributed by lot to the various contributing countries.

Forty-two kilogrammes were constructed. Their densities were

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determined by the most accurate processes, the weighings being made in a balance especially constructed for this work, which was capable of weighing *in vacuo*, although the actual weighings were made in air. The actual construction of the balance permitted the transposition of the standards and a manipulation of the weighings without approaching the balance nearer than about 4 meters.

An analysis of the material of which these standards are composed, showed that it was almost exactly in accordance with the original plan, that is to say, that it contained 90 per cent. platinum and 10 per cent. iridium. There were traces, however, of rhodium and iron. The two meters, which came to the United States in the allotment, were Nos. 21 and 27, and that of the alloy of 1874, is No. 12. Of the kilogrammes, we obtained Nos. 4 and 20. The equation of these standards is given below:

Equation of the United States Prototypes.

METERS.

Microns.

No. 21	=	M	×	2.5	+	8.665 T	+	0.00100 T^2
" 27	=	M	—	1.6	+	8.657 T	+	0.00100 T^2

Alloy of 1874.

No. 12	=	M	+	3.3	+	8.634 T	+	0.00100 T^2

KILOGRAMMES.

				Mg.	Density.	Volume, c.c m.
No. 4	=	K	—	0.0857	21.5436	46.4176
" 20	=	K	—	0.0398	21.5509	46.4017

These values of the national prototypes were determined at the International Bureau of Weights and Measures by a very elaborate series of comparisons between them and the international prototypes. In the above equations, the "micron" is the one-millionth part of a meter; T is the temperature in degrees Centigrade.

With regard to the cross-section of the meter, the form shown in Fig. 1, was selected as having the greatest rigidity for a given amount of metal, and at the same time because, allowing so much of its surface to be exposed, it was believed to be capable of assuming most readily the temperature of the surrounding air. The surface, upon which the defining lines are engraved, is the neutral plane of the bar.

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The following comparison between the new prototypes and the *Mètre des Archives*, sets forth the advantages of the former:

Rigidity.		Superficial Area of Transverse Section.	
New Prototypes.	Mètre des Archives.	New Prototypes. Sq. Mm.	Mètre des Archives. Sq. Mm.
25.9	1.	150	100

The construction of these standards was completed in the autumn of last year, and those allotted to us were received by Dr. B. A. Gould, our representative in the International Bureau, and were by him turned over to the United States Minister at Paris, Mr. Whitelaw Reid. Meter No. 27 and kilogramme No. 20, and also meter No. 12, of the alloy of 1874, were received from the United States Minister by Professor George Davidson, Assistant in the United States Coast and Geodetic Survey, who brought these standards from Paris to Washington, where they were deposited in the Office of Weights and Measures. The utmost care was necessary and was shown in the transportation of these standards, and I am glad to say that they were received at the Office of Weights and Measures in excellent condition. On January 2, 1890, the meter No. 27 and the kilogramme No. 20 were carried to the Cabinet Room in the Executive Mansion, where the ceremony of breaking the seals upon the boxes was performed in the presence of the President of the United States, the Secretary of State and the Secretary of the Treasury, together with a distinguished company of scientific men, including the presidents of the engineering societies of the country, and all leading exponents of the science of metrology.

A formal certificate, declaring the condition of these standards on the opening of the boxes, was signed by the President, and witnessed by the Secretary of State and the Secretary of the Treasury. A somewhat similar certificate was signed by all the other gentlemen present at the ceremony of breaking the seals. By this official act of the President of the United States, meter No. 27 and kilogramme No. 20 are placed in a rank above that of any other standards which we now possess, and they will, therefore, be guarded as our national prototype meter and kilogramme.

Meter No. 21 and kilogramme No. 4 are still in the possession of the United States Minister at Paris, and will be brought to this

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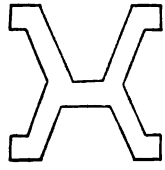
country by a special messenger from the Office of Weights and Measures during the coming summer.

In conclusion, I beg to call attention to the models of the new standard meter and kilogramme, which I have upon the table before me. The model of the meter is made of wood covered with aluminum, so that it resembles very much the platinum of the original, and as to dimensions, it is a very exact copy. It will enable you to see the form of the standard, and especially the interesting form of the cross-section which has been selected after much study and examination. Fig. 1 is a drawing of this section. The model of the

FIG. 1.



Cross-Section of United States Prototype Meter, 20 mm. wide and high (outside).

kilogramme is hollow, made of brass and plated with nickel, so as to resemble the platinum original. It resembles the latter, of course, only in form, but in that respect, is a very exact copy, and will enable you to see that this kilogramme is a cylinder, the diameter of which is equal to its altitude and the edges of which are very slightly rounded.

In thanking you for your kindness in granting me the privilege of discussing this question before you this evening, allow me to say, that the Office of Weights and Measures will always be pleased to do anything in its power to further the interests of exact measurement in any operations growing out of the practice of your profession. We can determine the errors of your standards of length, if you desire it, and of your standards of mass also. We have recently published a two-page table for the transformation of the customary system of weights and measures to the metric system, and I have placed a number of these upon the table, which I should be pleased to have any of you appropriate when an opportunity is offered, if you so desire.

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PRELIMINARY NOTE ON THE THERMAL PROPERTIES OF SLAGS.

BY HENRY M. HOWE, A.M., S.B., BOSTON, MASS.

(Washington Meeting, February, 1890.)

SECTION 1. INTRODUCTION.—In this investigation the melting-point and the specific heat of certain artificial silicates were determined by the method of mixtures, using a large platinum-ball pyrometer. The silicates were placed in platinum crucibles on the hearth of a small regenerative gas-furnace (Figs. 1 and 2), and the platinum ball was placed between them. The temperature of the furnace was usually raised very slowly, the behavior of the silicates being observed as they gradually grew hotter; and their temperature was ascertained indirectly from time to time by determining that of the platinum ball, which to that end was temporarily withdrawn from the furnace and immersed in a known weight of water in a calorimeter. The weight and specific heat of water and of platinum ball being known (the specific heat of the latter approximately), it was possible to calculate the temperature of the platinum ball, which was assumed to have been substantially that of the silicates which surrounded it in the furnace.

The specific heat of the silicates also was determined by the method of mixtures. Their temperature having been determined as just described, one of them was dropped into a known weight of water in a calorimeter, whose rise of temperature was then noted. Knowing the weight and initial temperature of the silicate, its specific heat was easily calculated.

The results obtained, especially as regards the melting-point, must be regarded as first approximations. The technical difficulties encountered were so serious as to raise doubts, now from one cause, now from another, as to the accuracy of most of the results: hence I give but a few, those which were reached without mishap, and with proper observation of the various precautions. The difficulties encountered are referred to again in Section 6.

The investigation was carried out in the metallurgical laboratory of the Massachusetts Institute of Technology, and part of the

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expense was met by a grant from the Elizabeth Thomson Science Fund. My hearty thanks are due to Professors R. H. Richards and S. W. Holman for invaluable advice and aid, and to Mr. H. A. Hastings, who, when a student, did a great deal of the work connected with the investigation, assisting me in a most laborious and painstaking way.

I intend to determine the specific heat of some silicates, after making certain important modifications of the method; and I hope to resume the study of their melting-points later.

SEC. 2. THE FURNACE.—It was necessary to have command of a very high temperature, at least 1400° C. (2552° F.), and to make such dispositions that the platinum ball, and the silicate or silicates experimented on, should be at approximately the same temperature at the moment of withdrawing the former. Several trials were made with a small Fletcher's gas-furnace, such as is used for dentists' work, and shown in elevation at A (Fig. 2), the platinum ball lying within a large platinum crucible, which also contained the silicate under examination, the platinum ball being kept out of actual contact with the silicate by enclosing it in an inner small platinum crucible.

While this apparatus served satisfactorily for certain preliminary work, its temperature was neither uniform enough nor under sufficient control. The regenerative gas-furnace, shown in elevation at B, Fig. 2, and in section in Fig. 1, was, therefore, used. The latter shows the form finally arrived at after several changes.

The furnace is of the reversing regenerative or Siemens type, with two regenerators, FF′, loosely filled with angular, irregular lumps of fire-brick. Through one of the regenerators at a time (say F) part of the air used for combustion is brought under pressure from a blower, the products of combustion passing out through the other regenerator and through the pipe, G′, to waste. Common illuminating gas is used for fuel, and is brought in alternately through one (say J) of the gas-pipes, JJ′. With this gas is mixed a considerable quantity of air, brought alternately by one (say H) of the pipes HH′. It was found necessary to thus mix quite a large volume of air with the gas before admitting it into the furnace, to prevent rapid decomposition of the gas with deposition of carbon. But, in spite of this precaution, sufficient carbon appears to have deposited to injure the platinum vessels rapidly. At intervals, usually of five minutes each, the furnace was reversed by means of common three-way gas-cocks shown in Fig. 2.

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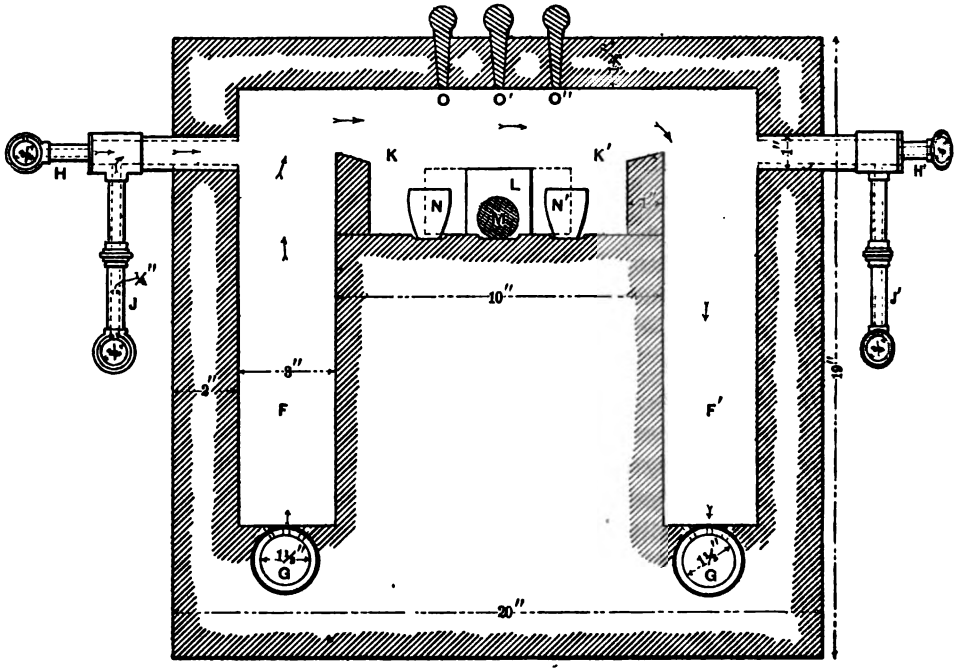
Although only part of the air and none of the gas was pre-heated in this furnace, a temperature of 1400° C. has been reached in it, and pure bisilicate of lime has been sintered together to a firm, stony mass, so firm that it could be dropped 2 or 3 feet upon a wooden floor without breaking or chipping. Lime-magnesia silicates have been completely melted.

The hearth of the furnace was made of a moulded brick, with

Fig. 1.



SIEMENS GAS-FURNACE.
Scale ⅛

depressions for five platinum crucibles, NN′, and for the platinum ball M. Crucibles and balls were introduced and removed through the doorway, L, closed with a tightly-fitting moulded wedge-brick. In removing them for calorimetric observations they were grasped in suitably shaped tongs, whose ends were previously heated to redness, to lessen the transfer of heat from the ball or the crucible to the tongs.

OO′O″ are peep-holes for examining the contents of the crucibles. At temperatures at and below moderate whiteness their condition could be seen; at higher temperatures the eye, even when thoroughly

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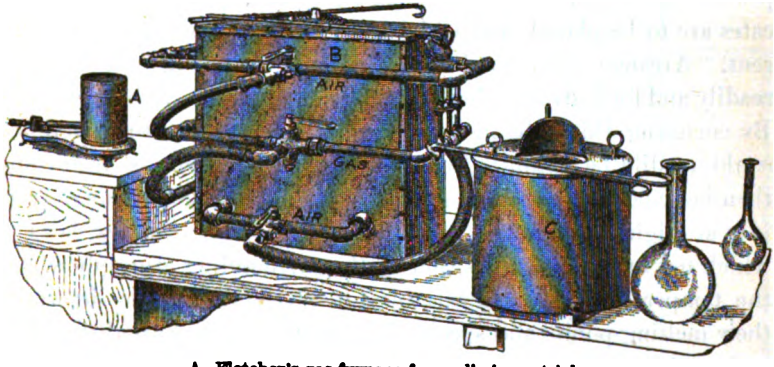
protected with dark glasses, so that it was not at all dazzled, could detect nothing or next to nothing. This peculiar result is due to the nearly uniform high temperature of the objects observed, closely approximating, after equilibrium has been attained, the temperature of the flame. As a consequence, all points glow alike and reflect alike, presenting to the eye none of those differences in tint, shadow or luminous intensity by means of which separate forms might be discerned or inferred. Single luminous points or lines, like the electric arc, or luminous surfaces not uniform, like that of the sun, can be easily examined through dark glasses and distinctly seen.

In order to learn the condition of the silicates at high temperatures it was found convenient to probe them with stout platinum wires let into the ends of iron rods, a separate rod and wire being

Fig. 2.



A. Fletcher's gas-furnace for preliminary trials.
B. Siemens' gas-furnace.
C. Calorimeter.

provided for each silicate. As long as a silicate was unaffected, nothing, or at most a light powder, adhered to the wire. When it became sintered together nothing adhered to the wire, and I could sometimes recognize by the feeling, or by the appearance of the slag, that it was sintered. When the slag was pasty the wire would draw strings of it out. When it was more or less completely molten a bead was found at the end of the wire.

Deterioration of Platinum.—The platinum crucibles, and in a lesser degree the platinum ball, deteriorated very rapidly when used in the gas-furnace, growing brittle. This is probably due to absorption of carbon from the flame of the furnace, a certain amount of carbon-deposition occurring as the gas, entering the furnace, became rapidly heated.

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Several ways of avoiding this trouble suggest themselves: (1.) Limiting more carefully the proportion of gas to air admitted. (2.) Mixing gas and air more thoroughly before admission. (3.) The use of hydrogen, carbonic oxide or water gas, instead of coal-gas. (4.) Mixing a little oxygen with the blast, so that a more strongly oxidizing flame may be obtained. Were we to make the flame much more oxidizing by simply increasing the proportion of air to gas admitted, we could not develop a high enough temperature in the furnace. (5.) Highly pre-heating the coal-gas before admitting it into the furnace, *e.g.*, by passing it through incandescent coke, in which the carbon set free by its decomposition would be deposited, as on a filter.

It is felt, however, that an electric-furnace should give better results. Several forms have been under consideration, of which the most promising seems to be a metallic or carbon tube in which the silicates are to be placed, and which is to be heated by an electric current. Arrangements for introducing and removing the silicates readily and for learning when they become molten have been devised. By enclosing this tube in one of glass or porcelain the operation could readily be conducted *in vacuo*, and the silicates of iron could then be experimented on. As their state of oxidation changes rapidly at high temperatures, and according to very complex laws, in which not only the composition of the surrounding atmosphere, but the temperature also, are variables, it is very difficult to determine their melting-points and specific heat under common conditions.

SEC. 3. CALORIMETER (Fig. 3).—The principal points to be kept in view, in designing a calorimeter for the method of mixtures, are that the calorimeter, its contents, and the hot substance (*e.g.*, the platinum ball) whose temperature is to be determined, may quickly reach a common temperature; that this temperature, and from it the rise of temperature of the calorimeter, may be measured accurately; and that the transfer of heat from the calorimeter to the air of the room and back may be accurately known.

In order that the temperature of the calorimeter and that of its contents, in this case water, may rise together, so that when the rise of temperature of the water is determined, that of the calorimeter which contains it may also be known, this vessel should be extremely thin and of high thermal conductivity. Hence the employment of very thin sheet brass. The brass is actually 0.0066 inch (0.000,168 m.) thick, but, thanks to some angle-pieces, it is stiff enough. The weight of the part below water, including the stirring-pan and

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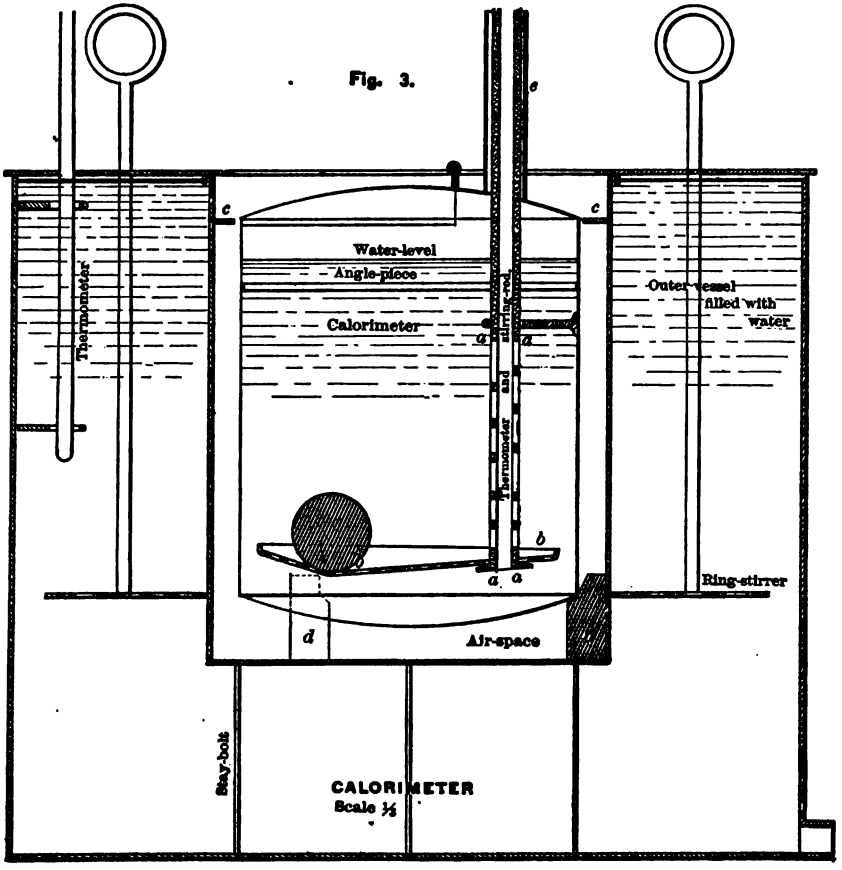
rod, *a* and *b*, is only 193.95 grammes. Assuming that the part below water reaches the temperature of the water itself, and that the temperature of the part above water, including the lid, rises during an observation half as much as that of the water, the thermal capacity of the calorimeter per 1° C. is as follows:

The part below water,	18.23	calories.
The part above water,	2.64	"
The thermometer,	0.75	"
The water (1.5 litres),	1500.00	
	1521.62	

Fig. 3.



As the thermal capacity of the water is thus 98.58 per cent. of the whole, any error introduced by assuming incorrect absorption of heat by the different parts of the vessel is insignificant.

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In order to measure the rise of temperature accurately, a thermometer reading easily to 0.01° C. is used. It stands within the perforated brass tube, *a a a a*, which serves also as a handle for stirring. By it the perforated brass dish, *b b*, and the platinum-ball or other hot object lying thereon, are raised and lowered. The thermometer hangs freely in this tube and rises and falls with it, care being taken that the mercury does not rise to the surface of the water.

In order that the transfer of heat, back and forth from the calorimeter to the surrounding objects and *vice versa*, may be accurately known, it should be small. In order that the radiating outer surface may be small, the calorimeter is a short, stout cylinder with dished ends, as near an approach to the spherical form as is convenient; and its surface is nickel-plated and highly polished, that it may radiate but little heat. It is surrounded by an air-space, which is partly closed at the top of the ring, *c c*. This permits air to circulate, so that the air in the air-space may be nearly at the temperature of the room; but it prevents rapid movements of air, which might remove heat at an irregular rate from the outside of the calorimeter. As *c c* is practically at the temperature of the outer vessel, which is very nearly the initial temperature of the calorimeter, the radiation of heat to it from the walls of the calorimeter is more regular than the radiation to external objects would be were *c c* omitted.

The calorimeter rests on three vulcanite stools, *d*, so shaped that the surface of contact is extremely small, just enough to keep the calorimeter from tipping over during stirring.

Again, the handle by which the stirrer is raised and lowered consists of a vulcanite cylinder, which also surrounds the thermometer, preventing the heat of the fingers from being conducted to the stirrer and thermometer, and preventing the heat radiated from surrounding objects from striking the thermometer. A slit is left in the cylinder for reading the thermometer, and, indeed, the upper part of the stem, where rapid readings are made, projects above the vulcanite cylinder.

As the stirrer is raised and lowered, some parts of its stem are at one time below the surface of the water, at other times above the upper surface of the calorimeter. From these wet surfaces water evaporates rapidly; so that on the down stroke the stem, thus cooled, in turn absorbs heat from the water, introducing an element of error. To guard against this a very light sheet-brass tube, *e*, is

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fastened to the cover of the calorimeter, surrounding the stem of the stirrer, which, however, plays freely in it. This tube prevents draughts of air from striking the moist part of the stirrer stem, and so far diminishes the loss of heat from this source that its variations are insignificant, as is shown by the smoothness of the cooling curves.

In order further to diminish the transfer of heat from and to the calorimeter, some experimenters have arranged that the temperature of the surrounding air shall be about midway between the initial and final temperatures of the calorimeter itself, and have surrounded the calorimeter with non-conducting substances, such as eider-down. But it is clearly more important that this transfer be accurately known than that it be small, and the former of these two expedients introduces an element of uncertainty. Moisture condenses to an indeterminable extent on the outer surfaces of the calorimeter, if its initial temperature be below the dew-point of the surrounding air, to be evaporated when the calorimeter later becomes warmer than the surrounding air. To avoid this I always took care to have the calorimeter initially a little warmer than the air of the laboratory which, indeed, was always pretty dry.

An objection to the use of non-conducting coverings around the calorimeter is, that the transfer of heat is in this case liable to vary to an unknown degree; and though these variations may not be severe, it is as well to eliminate them almost wholly by surrounding the calorimeter with a body to which it will radiate heat at a nearly constant but very low rate. Actually an outer vessel filled with water was used, surrounding the calorimeter on all sides except the top, and separated from it by a half-inch air-space. By having the initial temperature of the water in both vessels approximately equal, by means of the air-space, and by nickel-plating and highly polishing the exterior of the calorimeter and the inner surface of the outer vessel, the transfer of heat was kept low; while, thanks to the great quantity of water in the outer vessel, its temperature was affected but slightly by the rise of temperature of the calorimeter proper. In a direct experiment, on filling the calorimeter with water as much hotter than that of the outer vessel as the calorimeter would become in actual use, the temperature of the outer vessel was not affected. The details follow.

Experiment 5, January 6, 1888. Does the rise of temperature of the calorimeter raise the temperature of the outer vessel sensibly?

Placed 14 liters of water at 20.99° in the outer vessel. Then placed 1.5 liters of warm water in the calorimeter.

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Time.			Temperature of outer vessel.	Temperature of water to be placed in calorimeter.	Remarks.
H'rs.	Min.	Sec.	Degrees.	Degrees.	
3	30	25	20.99	.....	
3	31	20	20.99	.....	
3	33	40	.....	32.42	
3	34	25	.....	32.48	
3	37	25	.....	32.23	
3	39	00	.....	.....	Began pouring water at 32° into calorimeter.
3	39	12	.....	.....	Finished do. do.
3	40	00	20.98	.....	
3	40	25	20.98	.....	

Thus the temperature of the outer vessel, far from being raised by heat radiated to it from the calorimeter, fell slightly, probably owing to evaporation.

A further advantage of surrounding the calorimeter with a body whose temperature is either equal to or very slightly below the initial temperature of the calorimeter itself is, that the correction to be applied on account of loss or gain of heat by the calorimeter itself during an experiment, *i.e.*, *the cooling-correction*, is found by a comparatively simple procedure.

The cooling-correction is found as follows: The temperature of the water in the calorimeter and of that in the outer vessel being brought to nearly the same point, slightly above that of the air of the room, the thermometer within the calorimeter is read repeatedly. It should show a practically uniform temperature. The hot substance whose temperature is to be determined, *e.g.*, the hot platinum ball, is now dropped into the calorimeter with as little splashing as possible, being caught on the perforated dish, *b b*, which to that end has been brought near the surface of the water, but is immediately lowered on receiving the hot substance, and now rapidly raised and lowered during the remainder of the observation by means of the perforated brass-tube handle, *a a*. Care is taken not to raise the hot object so near the surface of the water that the steam which it forms would escape uncondensed, as this would cause loss of heat to an indeterminable extent. The cover of the calorimeter is, of course,

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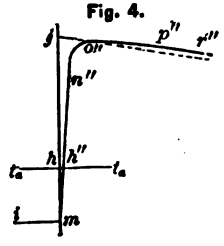
tightly closed the instant that the hot substance is dropped in. This instant of immersion is noted very carefully.

While stirring, the time and the corresponding temperature of the calorimeter are recorded very often. The temperature at first rises very quickly, owing to the transfer of heat from the platinum ball to the water. It soon reaches a maximum and again declines, though very slowly, the loss of heat by evaporation and by radiation from the outside of the calorimeter now exceeding the transfer of heat from the ball to the water, which, of course, diminishes asymptotically as the temperatures of the ball and water approach each other.

Fig. 4.



These observations are plotted with temperature as ordinate, time as abscissa. A curve like $m\ n''\ o''\ p''\ r''$, Fig. 4, results, m representing initial time and temperature, o'' maximum temperature and corresponding time. A straight line, $j\ o''$, is drawn parallel with $p''\ r''$; j is the maximum temperature that the water would have reached had there been no loss of heat, and $j\ m$, is the corrected rise of temperature of the water.

The smoothness of the curve $m\ o''\ p''\ r''$, depends on the frequency and accuracy of the readings, and more especially on the care used in stirring. After a little experience I obtained curves so smooth as to leave nothing to be desired. One is shown in Fig. 5.

The more rapidly the hot substance gives off its heat to the water, the smaller, of course, is the cooling-correction. In case of the platinum ball, the temperature of the water rose very suddenly to a maximum, thanks to the high thermal conductivity of the platinum. In ten cases taken at random, I find that the maximum was reached in from thirty-three to fifty seconds from the time of immersion. The cooling-correction was, in general, 0.02° C. in case of the platinum ball, in only one instance rising to 0.04° C. So, too, when small pieces of slag were immersed when at an only moderately high temperature, the maximum was reached in about one minute, and the cooling-correction was about 0.03° C.

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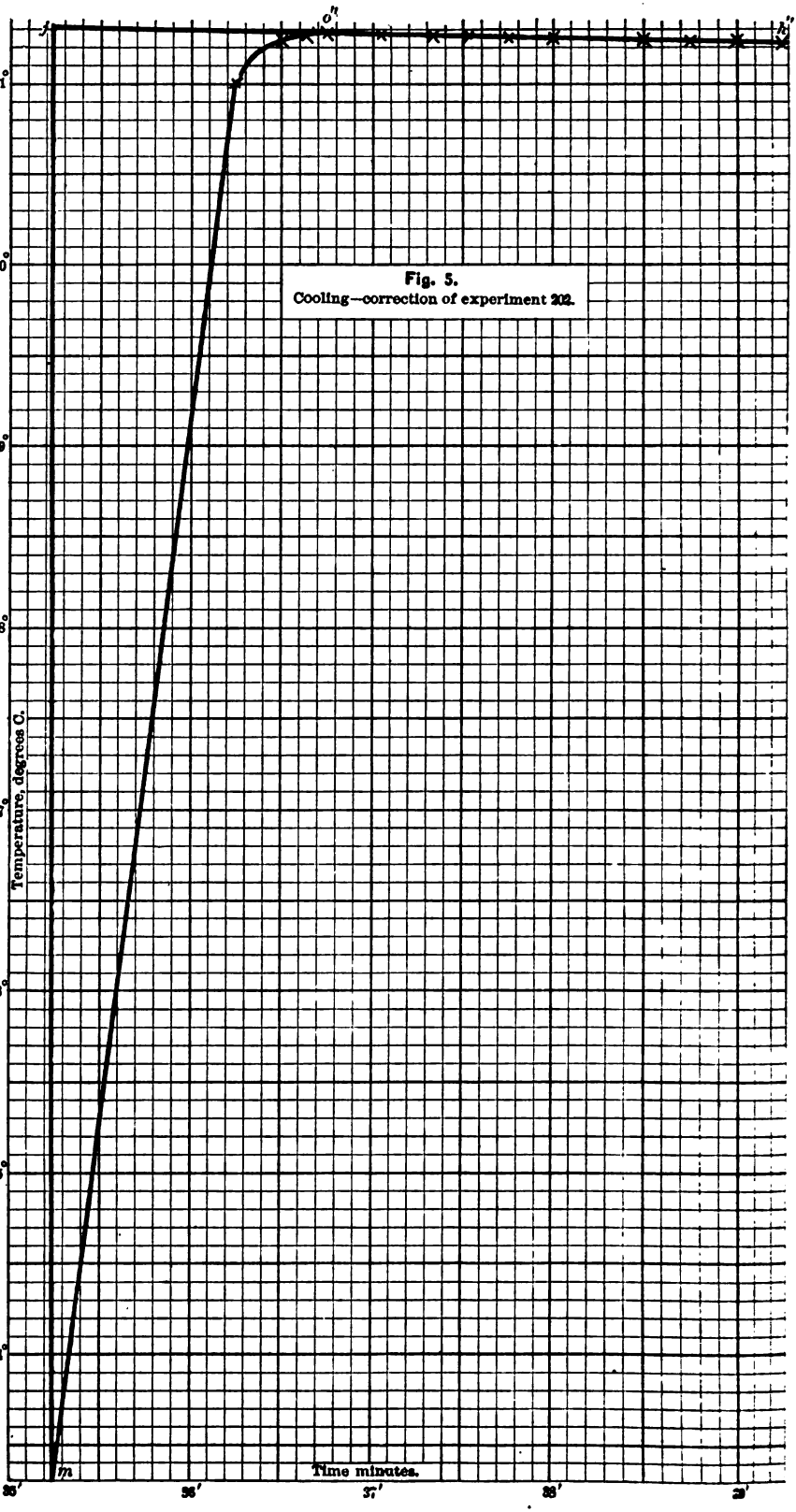

Fig. 5.
Cooling—correction of experiment 202.

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If, on the other hand, a large block of slag was immersed when very hot, and if it did not break up in the water, but held together in a single piece, it gave off its heat so slowly to the water that three or even four minutes elapsed before the maximum temperature was reached; the temperature remained stationary at the maximum sometimes for more than a minute, and five minutes more might elapse before the temperature fell 0.06°. In one case the cooling-correction yielded by a very regular curve amounted to 0.20°.

While it is not necessary, therefore, to plot the cooling curve for every observation of temperature made with the platinum ball, the cooling-correction being safely inferred from the uncorrected rise of temperature, I deemed it necessary to plot it for every determination of specific heat of a silicate, since here the rate of transfer of heat from silicate to water, and the consequent cooling-correction, depended not alone on the temperature, weight, thermal conductivity and specific heat of the slag (four variables, while in case of the platinum ball we have but one, the temperature), but also on the size of the fragments into which the slag was broken by the sudden cooling action of the water.

Size of Calorimeter and Platinum Ball.—The radiation of heat from the platinum ball during its transfer from the furnace to the calorimeter must be rapid, owing to the very high temperature employed, rising to 1400° C. It would be difficult to prevent more or less conduction of heat from the ball to the tongs during this transfer. Lining the tongs with brick makes them unwieldy. If simply faced with asbestos or clay, more or less of this substance is liable to attach itself to the platinum ball, or to fall directly into the calorimeter, creating a source of error. If the tongs are either of naked platinum or of iron, they must be much cooler than the platinum ball, lest they weld to it, a difficulty which often confronted me. At one time I wrapped the platinum ball in a platinum-wire netting, with a loop which I caught in the tongs, thereby lessening the conduction of heat from ball to tongs. But I abandoned this later for three reasons: 1st. Because a second or more was often lost in finding this loop owing to the invisibility of all matter within the furnace, while the ball itself could be grasped very rapidly, as I knew its position in the furnace, and could seize it even if I could see it only very faintly. 2d. Because the extended surface of the netting increased the radiation of heat. 3d. Because I believed that little, if any, more heat escaped from the platinum ball when grasped directly in broad red-hot tongs, which, though touching it only at two points,

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surrounded and shielded it to a considerable extent, than when it was carried naked through the cool air of the laboratory.

The time occupied in transferring the platinum ball from the furnace to the calorimeter was twice found to be 1.6 seconds.

In order that the loss of heat during this 1.6 seconds might be but a trifling proportion of the total heat of the platinum ball, and in order that the temperature of the ball itself might be only slowly affected by the brief local variations or flutterings of temperature in the furnace, and might thus be very closely the temperature of the silicates under observation, the ball was made large. It weighed at different times from 292.8 to 290.88 grammes, and later, after repairs, 262 grammes.

The composition of the platinum ball has not yet been determined. If it is later found to contain an important quantity of foreign elements, it may be necessary to apply a slight correction to the results obtained.

To lessen the loss of heat during the transfer from the furnace to the calorimeter, Pionchon carried the hot substance in a lime-lined crucible, and, having brought this close to the calorimeter, removed its lid, and then transferred the hot substance from it to the calorimeter by means of pincers and platinum wire. It took about three seconds to carry this crucible from the furnace to the calorimeter, and a fraction of a second to transfer the hot substance from the crucible into the calorimeter. The loss of heat during the transfer in this method is doubtless somewhat less than in mine; on the other hand, it must take very much longer to bring the hot substance to the temperature of the furnace, and, unless this were almost constant, serious errors might arise. This danger would be especially great in determining the specific heat of silicates, whose thermal conductivity is so much less than that of platinum. I have devised a modified plan which should avoid this danger.

The size of the calorimeter, *i.e.*, the quantity of water it must hold, should be such that its rise of temperature during an observation may be neither too great nor too small. Not too great, lest the water grow so hot that heat radiates from it rapidly; not too small, lest the errors in reading the thermometer (due mainly to parallax) bear a considerable ratio to the rise of temperature itself.

Again, the calorimeter must not be too small, lest the heat radiated from its surface bear too great a ratio to the total quantity of heat which the water within it takes up from the platinum ball or other hot object within it.

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The calorimeter was actually designed to hold 1.5 litres of water, and its temperature is raised about 10.5° C. when the platinum ball at 1400° C. is placed within it. In reading to 0.01° C., the thermometer thus reads to within 0.1 per cent. of the total rise of temperature of the water in the calorimeter.

SEC. 4. FORMULÆ.—(A.) *To Find the Temperature of the Platinum Ball.*—The quantity of heat given up by the ball to the water in the calorimeter equals that needed to heat the latter from its initial to its corrected maximum temperature.

If w = the weight of the ball,

S = the specific heat of platinum, assumed for the moment to be constant,

W = the total water-equivalent of the calorimeter, stirrer and water,

i = the initial temperature of the calorimeter,

m = the maximum temperature of the calorimeter,

T = the initial temperature of the ball; then

(1.) $w \cdot S\,(T - m) = W(m - i)$, and

(2.) $$S \cdot T = \frac{W(m-i)}{w} + S \cdot m.$$

But the specific heat of platinum varies so much in the range of temperature covered by the proposed investigation that we must take its variations into account.

If, now, we let

S_T = the mean specific heat of platinum from 0° to T°,

S_m = the mean specific heat of platinum from 0° to m°,

then equations (1) and (2) become

(3.) $w(S_T T - S\,m) = W(m - i)$, and

(4.) $$S_T T = \frac{W}{w}(m - i) + S_m m.$$

The values of S_T and of S_m may be found by the formula:

(5.) $k_t = 0.03208 + 0.000{,}005{,}44t + 0.000{,}000{,}000{,}016t^2$.*

Pionchon adopts the formula:

$k_t = 0.0317 + 0.000{,}006t$.

The variation in S_m, in the range of temperature covered, is so

* Prof. S. W. Holman arrives at this formula by discussing the results of Violle, Pouillet, Byström and Weinhold. While we may not regard it as final, it is at least a close approach to the truth; and results reached by means of it may be corrected later, should further researches show that it is appreciably inaccurate. At any rate, it yields us results which are closely comparable among themselves.—J. C. Hoadley, *Journ. Franklin Inst.*, August, 1882.

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slight that we commit no sensible error in taking it always as the mean specific heat of platinum from 0° to 29°, or 0.032,238.

$\frac{W}{w}$ being also constant, the only variables are S_T, T, m and i. In this form the observations may be reduced rapidly by means of a curve platted with $S_T T$ as ordinate and T as abscissa, the values of S_T being calculated by formula (5).

B. *To Find the Mean Specific Heat or Total Heat of Fusion of a Silicate.*—The heat given out by the silicate in cooling from its initial temperature to the corrected maximum temperature of the calorimeter, equals that needed to heat the calorimeter from its initial to its corrected maximum temperature.

If we let

w' = the weight of the silicate,

S' = its mean specific heat between T' and m,

T' = its initial high temperature, then

(6.) $w'S'(T'-m) = W(m-i)$, and

$$S' = \frac{W(m-i)}{w'(T-m)}.$$

If the silicate was at its melting-point and molten, then its total heat of fusion from $m°$ is,

$$(7.)\ S'(T'-m) = \frac{W(m-i)}{w'}.$$

SEC. 5. APPLICABILITY OF THE METHOD.—For determining high temperatures, though accurate and trustworthy, the method of mixture is rather inconvenient, laborious and expensive; inconvenient, chiefly because of the length of time which must elapse between successive determinations of temperature. I believe that Le Chatelier's thermo-electric pyrometer would be far more convenient.* Still, the platinum-ball pyrometer can well be used to check the thermo-electric one from time to time.

Another method of determining temperatures and melting-points which has become prominent lately, is Seger's. He introduces into the furnace or other place whose temperature is to be determined, a series of slender fire-clay pyramids, of different composition and known softening-points, and notes which of these soften and which do not. But unless an inconveniently large number of pyramids be

* *Journal de Physique*, 2d series, vi., Jan., 1887; *Génie Civil*, March 5, 1887. "Étude sur la mesure des temperatures élevées, et sur le pyromètre thermo-électrique de M. le Chatelier," by Emilio Damour.

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used, the points on his scale must be far apart, and interpolation between them must be difficult.

For determining specific heats, the method which I employed would be satisfactory for technical purposes, were the temperature of the furnace determined by Le Chatelier's or some other rapid method, and were errors from the fourth of the following sources better guarded against.

SEC. 6. DIFFICULTIES AND SOURCES OF ERROR.—The following are the chief sources of error noticed:

1. As the components of the silicate should be very finely powdered before mixing them, a small quantity may be blown out of the crucible by currents of air before they are sintered. Should more of one than of another be blown out, the composition of the silicate would change. This may be met by binding the mass together with a little dextrine; or, better, in case the substance need not be under observation, by enclosing it wholly in a platinum envelope.

2. Little particles of brick may fall from the roof of the furnace or be carried from the regenerators, sides, hearth, etc., into the crucibles by the draught, changing the composition of the silicates under observation.

3. The temperatures may not be the same in different parts of the furnace, owing to irregular conduction of heat through its walls, indraughts of air, etc.

4. Heat may be lost in transferring the hot substances from the furnace to the calorimeter.

5. The composition of the silicates may be changed by the absorption of sulphur, and in case they contain iron by the absorption of oxygen, from the furnace gases, and possibly by the absorption of nitrogen or hydrogen.

6. The almost complete invisibility of the matter within the furnace, already dwelt on, is a source of great difficulty, if not error.

7. At high temperatures the stickiness and softness of the platinum vessels is most inconvenient.

8. At least twenty minutes, and preferably half an hour, should, I think, elapse between successive determinations of temperature by the platinum ball. This places the method at a great disadvantage compared with that of measuring the temperature by electrical resistance (Siemens's) or thermo-electric power (Le Chatelier's), which allows us to read the temperature continuously.

9. In determining specific heats, we have to determine the heat

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in the platinum ball and in the other hot substance separately. As I used but one calorimeter, several minutes had to elapse between these observations. The temperature of the furnace was liable to change in this interval, and it was especially likely to be lowered on opening the door for removing the first of the two hot substances. According to whether the platinum ball or the other hot substance were removed first for calorimetry, the observed specific heat would be too low or too high. That the error from this source is considerable, is indicated by the following data:

In experiments 22 and 23 blocks of the same slag were experimented on. That of experiment 22 was removed for calorimetry 12′ before, that of 23, 7′ 40″ after the removal of the platinum ball; 22 gave the specific heat 0.13 higher than 23.

Again, the steel ball of experiment 21 was removed for calorimetry 10′ 15″ after that of experiment 20; the specific heat was 0.00435 greater in the first than in the second. The steel ball of experiment 20 was removed 9′ 25″ after the platinum ball, and its specific heat is slightly lower, to wit, 0.00164 lower, than Pionchon's formula for iron implies.

In addition to these sources of error, which arise from the particular methods employed, we have two more which are inherent, and hence inevitable.

10. The acid-earthy silicates have no well-marked melting-point, but pass very gradually from the solid to the molten state, through a considerable range of temperature. Some given degree of fluidity, then, must be chosen quite arbitrarily as corresponding to the melting-point.

Even if we seek not a sharp melting-point, but rather a "melting-range," *i.e.*, the range of temperature between complete solidity and complete liquidity, the upper and lower limits of this range, if they are to be determined accurately, must be recognized by some arbitrary standard of viscosity. There seems to be no distinct critical point at which viscosity begins or ends.

But while different observers might choose widely different standards for the melting-point, which, according to their point of view, might be anywhere between complete solidity and complete fluidity, the differences of opinion as to where viscosity begins and ends would be much less important.

11. It is very possible that, at least in certain cases, the temperature required to cause the components of a given mixture, even if finely ground and intimately mixed, to unite and melt is much

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higher than that at which the resulting silicate will again melt. At the melting-point of lime-bisilicate, for instance, lime and silica as such are quite solid. Now before the silicate can melt, the solid lime and solid silica must combine with each other; and there is no necessary relation between the temperature at which they can react on each other and combine, and that at which the product of their union when formed will melt. Calling these temperatures T_c and T_f, T_c may equal, or it may be higher or even lower than T_f.

Now it is generally believed that T_c is usually higher than T_f.* If this be true, then we should discriminate carefully between T_f, which is the melting-point proper, and T_c. For most metallurgical purposes it is more important to know T_c than T_f.

But even if T_c is no higher than T_f, one would still expect that, in order to melt a given mixture for the first time, we would have to heat it well above T_f.

Assume for the moment that we have heated lime and silica, previously finely ground and intimately mixed in proportions corresponding to lime-bisilicate, to the melting-point (T_f) of this substance; and, further, that this is the most fusible silicate of lime. Now we would expect that only those molecules of lime which actually touch silica could unite with it. But, however fine the grinding, only an insignificant proportion of the molecules of a given mote or speck of silica will be at the surface of that mote, the rest occupying its interior, and only this insignificant proportion will at first touch base. Now if the temperature be only barely at the melting-point of lime-bisilicate, and by assumption below that of all other possible products, the little molecules of lime-bisilicate thus formed can hardly unite with the adjoining molecules of lime on the left and of silica on the right. At least if they united with them the products would hardly melt, for we would have at the left a silicate more basic and at the right one more acid than lime-bisilicate, and both would be infusible at the existing temperature.

If the mixture were not in the ratio of greatest fusibility the case

* It has always seemed strange to me that Percy held the opposite view, saying that it is probable that the components of a silicate will originally fuse, when they first enter into combination, at the melting-point of the silicate itself, provided sufficient time be allowed (*Metallurgy, Fuel*, p. 85, 1875). Ledebur, on the other hand, says without qualification, that the melting-point of a silicate once formed is often or usually below its formation-temperature (*Handbuch der Eisenhüttenkunde*, p. 150, 1884). Åkerman, too, says that the melting-point of an already formed slag is doubtless below the point at which a mere mechanical mixture of its components will fuse. (*Stahl und Eisen*, X., p. 425, 1890.)

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would be less simple. But still the fact that the mixing could never be uniform and effective except at the exterior of the individual particles, leads us to expect that it is necessary to heat a mixture well above T_f in order to melt it for the first time.

But even if we concede, for argument's sake, that a mixture will melt at T_f for the first time, it is extremely probable that when we heat it to T_f some considerable time must elapse before the molecules in the interior of the particles of base can unite with those in the interior of the particles of silica and melt. Assuming this to be true, it follows that, if we raise the temperature of a mixture continuously, it will actually reach a temperature above T_f before melting; and further that, if the temperature of two like mixtures were raised continuously but at different rates, the more slowly heated would actually melt at a lower temperature than the other.

It is further extremely probable that, if one of two otherwise like mixtures were less finely ground or less intimately mixed than the other, and if the temperature of both were then continuously raised at the same rate, the more finely ground or more intimately mixed would melt sooner, and hence at a lower temperature than the other.

In short, it is extremely probable that the point at which a gradually heated mixture melts depends not alone on its composition, but also on how fast it is heated, on how finely it is ground, and on how uniformly it is mixed.

The error due to the formation of steam, which other observers have had to guard against, was hardly appreciable in these experiments, because the hot substance was so hot that the water would not wet it at the moment of its immersion in the calorimeter, and it was so large that, by the time that it had cooled so far that the water could touch it and become vaporized, it had sunk so far that the steam formed was wholly condensed in passing up through the overlying water.

Most of these sources of error were foreseen; to guard against them fully would have introduced some additional difficulties and complications. It seemed best before introducing these to become familiar with the difficulties which necessarily arise in measurements at high temperatures, and thus to proceed from the simple to the complex.

The consideration that the inevitable error due to the 10th and 11th of the causes just noted was likely to vastly outweigh those due to the others, favored this decision.

SEC. 7. RESULTS.—The investigation, unfortunately, was chiefly

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directed to the determination of melting-points, before I had satisfied myself that it was less fitted for this than for determining specific heats. And much labor was fruitlessly put into an attempt to determine the melting-points of iron silicates, endeavoring vainly to prevent their degree of oxidation from changing. Still other efforts were made to determine the specific heat of iron, before learning that Pionchon had made known the remarkably complex law which it follows.* After his results mine are superfluous.

Thus it happens that the expenditure of much labor has not brought me far beyond the threshold of practically useful work. Indeed, I publish this note not to give my scanty results, but to furnish the details of the method, and point out the sources of error.

The more important materials used were white Berkshire sand, containing 99.69 per cent. of silica, and white Carrara statuary marble, containing 0.22 per cent. of insoluble matter (both by Drown's analysis).

A. *Specific Heat.*—The results of some of my determinations of specific heat are given in Table I.

TABLE I.—SPECIFIC HEAT.†

No. of Experiment.	Substance.	MEAN SPECIFIC HEAT.			
		From	To	Specific Heat.	Mean.
15	Steel	844°C.	30.86° C.	0.1748	
20	"	1022 "	34.14	0.1727	
22	Lime-iron singulo-silicate, $C_2 F_2 Si_1$ [4 CaO, 6 FeO_x, 5 SiO_2]	1058 "	27 "	0.262	0.255
23		1058 "	24 "	0.249	
26	Clay fire-brick	979 "	31 "	0.274	0.275
28	" "	1282 "	34 "	0.276	

Discussion.—These results harmonize tolerably with other data. Thus, Gruner found that the mean specific heat of fire-brick between 0° and 1100° was 0.26.‡ The difference between his result and

* *Thèses Presentées à la Faculté des Sciences de Paris*, par M. Pionchon, p. 43, 1886. Also *Comptes Rendus*, cii., p. 1455, 1886.

† The errors in the method tend to bring the obtained specific heats rather below than above the true specific heat.

‡ *Traité de Metallurgie*, i., p. 382, 1875.

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mine is not greater than may be reasonably referred jointly to experimental error and to difference in the composition of the brick treated.

Schinz found that the mean specific heat of four blast-furnace slags was as follows:*

Mean specific heat (from 0°C?) to.....	1000°C.	1200°C.	1300°C.
Maximum.....	0.2595 "	0.2846 "	0.2971 "
Minimum.....	0.2368 "	0.2580 "	0.2681 "

In order to compare my results on the specific heat of iron with Pionchon's, we must use, in calculating the results from my data, the value for the specific heat of platinum which Pionchon employs. On doing this, and calculating from Pionchon's formulæ the specific heat of iron in the same ranges of temperature, we get the following results:

Expt. No.	Mean Specific Heat of Iron.				Difference.	Difference ÷ mean.
	From	To	Pionchon.	Howe.		
15	837.58° C.	30.86° C.	0.17375	0.17636	+ 0.00261	1 ÷ 67
20	1013.7 "	34.14 "	0.17800	0.17636	— 0.00164	1 ÷ 108

The result of this comparison agrees with what we should expect from the number and kind of sources of error, to wit, that under the existing conditions the method should yield results near enough to the truth to be very useful for technical purposes, but not near enough for philosophical purposes.

B. *Melting-Points.*—Some of my results on this point are given in Table II.

Discussion.—In Table II. the silicates are arranged in what seemed to me their order of fusibility, taking into account their behavior both when melting for the first time, and when again setting, or remelting. Comparing this order with the order of the total heat of solidification, as found by Åkerman† and given in the last column,

* *Researches on the Action of the Blast-Furnace*, Maw and Müller's English Translation, 1870, p. 46.

†*Om Värmebehofven för Olika Masugnslaggers Smältning*, Pl. III., 1886. See also *Journal Iron and Steel Inst.*, 1886, i., p. 310.

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we find that there is a rough correspondence, quite as close as I expected to get, in view of the sources of error and uncertainty already set forth. Indeed, we need not expect a complete correspondence

TABLE II.

FUSIBILITY OF LIME-MAGNESIA SILICATES.

FIRST APPROXIMATION.

Formulæ.		Composition. Equivalents.			Ratio $\frac{RO}{100 RO}$		Highest Temp. (Cent.) at which Silicates were found.					
										Fully Molten.		
Chemical.	Metallurgical.	Lime.	Magnesia.	Silica.	Lime.	Magnesia.	Unsintered.	Sintered.	Partly Molten.	Without previous heating to a higher temperature.	After previous heating to a higher temperature.	Melting Point after Plattner.	Total heat of Fusion. Akerman.
							BISILICATES.						
7 CaO, 4 MgO, 11 SiO_2	$C_{1.75}MSi_{5.5}$	1.75	1.	2.75	64.	36.	1152	1224	.....	1273	1225	.....	418
2 CaO, MgO, 3 SiO_2	C_2MSi_6	2.	1.	3.	67.	33.	1152	.....	1273	1326	1225	.....	390
3 CaO, MgO, 4 SiO_2	C_3MSi_8	3.	1.	4.	75.	25.	1152	1273	1290	1326	1225	.....	412
3 CaO, 2 MgO, 5 SiO_2	$C_3M_2Si_{10}$	1.50	1.	2.5	60.	40.	.....	.....	.....	1273	.....	.....	431
CaO, MgO, 2 SiO_2	$CMSi_4$	1.	1.	2.	50.	50.	.....	1193	.....	1273	.....	2219	455
4 CaO, MgO, 5 SiO_2	C_4MSi_{10}	4.	1.	5.	80.	20.	.....	.....	1225	1326	1257	.....	435
CaO, SiO_2	CSi_2	1.	0.	1.	100.	00.	.....	1381	.....	.....	.....	2262	About 480
							SINGULO-SILICATE.						
CaO, MgO, SiO_2	$CMSi_2$	1	1	1	50	50	.....	1300	.....	.....	.....	.....	.....

between the total heat of solidification and the melting-point, for the former is a function not only of the melting-point but of two other variables—the specific heat and the latent heat of solidification.

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If we number the slags of Table II. in the order of their total heat of solidification as found by Åkerman, 1 having the least, and arrange them in the apparent order of fusibility given in Table II., they stand 3, 1, 2, 4, 6, 5, 7. Here only the 3, $C_{1.75}MSi_{3.5}$, and the 6, $CMSi_4$, are misplaced.

Now, without meaning to suggest ever so faintly an error on Åkerman's part, for his results are so concordant as to inspire confidence even were that not already assured by the weight of his name, it strikes me that the total heat of solidification which he finds for the magnesia-lime silicate, CMS_2, is surprisingly high, when we remember how fusible this slag has been found by other observers. Thus Gruner states unreservedly that it is more fusible than the silicate C_2MSi_6,* though, according to Åkerman, the total heat of solidification of the former is 17 per cent. greater than that of the latter. Berthier, too, says that it melts easily.†

The order of fusibility of the silicates $CMSi_4$ and CSi_2 is that given by Plattner.

Turning now to the absolute temperatures, my results are far below those which Plattner obtained by indirect calculation. This, however, is to be expected, as Plattner's method could only be expected to yield relative results, and as his absolute results are clearly much too high.

Turning next to Gredt's‡ and Åkerman's results, a much closer correspondence might be hoped for than we find. I can make no direct comparison, for Åkerman determined only the total heat of solidification, while Gredt did not determine the softening-point of any of the silicates that I experimented on. But we can make an indirect comparison, because Åkerman determined the total heat of solidification both for Gredt's silicates and for mine. Such a comparison follows:

	Formation—Melting Point.	Total Heat of Solidification.
Gredt's most-fusible lime-alumina silicate	1410°C.	361 calories.
My bisilicates	1273 to 1326°	390 to 480 calories.

* *Traité de Métallurgie*, i., p. 182, 1875.

† *Traité des Essais par la Voie Sèche*, p. 395, 1848.

‡ *Stahl und Eisen*, ix., p. 759, 1889. Gredt's experiments were made with Seger's normal fire-clay pyramids.

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Thus, while Åkerman finds that my bisilicates have higher total heat of solidification than this of Gredt's, I find lower melting-points than Gredt, and this though I sought a higher degree of liquidity, and hence should have found a higher melting-point, than Gredt, who sought rather the softening- than the melting-point.

Åkerman, indeed, thinks Gredt's results too high, for they can be reconciled with his own only by assuming values for the specific heat which he thinks too low.* Åkerman is probably right, though we can hardly locate the sources of our discrepancies confidently yet.

CRYSTALLINE MAGNETITE IN THE PORT HENRY, NEW YORK, MINES.

BY JOHN BIRKINBINE, PHILADELPHIA, PA.

AN unusually large number of crystals of magnetite obtained in the exploitation of a deposit of magnetic iron-ore of remarkable purity, encouraged the preparation of this paper, and its presentation at this time, that the matter might go on record in the *Transactions*. To make this record more complete, a preliminary statement is offered concerning the district and locality in which the ore is found.

To the northeastern portion of the State of New York the United States is indebted for a large percentage of the iron-ore which has assisted in developing the resources and industries of the nation. As early as 1804, iron-ore was mined between Lake Champlain and the Adirondack mountains, and this district has since been an important and continuous contributor to our iron-ore supply.

The production of the Lake Champlain district for the last eleven years is as follows:

Years.	Gross tons.
1879,	420,341
1880,	531,000
1881,	637,000
1882,	675,000
1883,	500,000
1884,	504,894
1885,	379,077
1886,	588,752
1887,	768,852
1888,	789,419
1889,	750,230
Approximate total,	6,544,565

* *Stahl und Eisen*, vol. x., p. 424, May, 1890. From *Jernkontorets Annaler*, 1889, v. and vi.

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In a paper upon the "Prominent Sources of Iron-Ore Supply,"* I drew attention to the importance of the iron-ore deposits of this district; and I quote from that paper, supplementing the figures of production there given by adding the estimated output up to the close of 1889.

Some of the Lake Champlain deposits have been operated for nearly a century; and the total output of this district to and including the year 1889 cannot be determined, but it is estimated at 15,750,000 gross tons. The largest producers are the mines at Mineville, near Port Henry, New York, from which 9,530,000 tons are estimated to have been obtained.

The total product of the Lake Champlain district in 1888 and 1889 was obtained from the following deposits:

	1889.	1888.
	Gross tons.	
From mines near Port Henry, N. Y.. . .	379,644	419,009
From mines of Chateaugay Ore and Iron Co., .	250,417	250,500
From mines near Crown Point, ...	75,169	74,910
From other mines (estimated), ...	45,000	45,000
Total, ...	750,230	789,419

Concerning the magnetites of northern New York, Prof. J. C. Smock, says: †

"The Adirondack region, the great mountain plateau of northern New York, is bounded by the valleys of Lake Champlain on the east, of the St. Lawrence river on the north and northwest, of Black river on the west, and the Mohawk on the south. Its area has been estimated to be at least 10,000 square miles. Dr. Emmons, in his survey of the second geological district, described the rock formations of this territory as gneisses and hypersthene rock principally; and the former he regarded as the prevailing rock, excepting in a large triangular area in Essex county, where the outcropping rocks are hypersthene. The gneissic rocks resemble closely the rocks of the Highlands of the Hudson, and they have been recognized by geologists generally as the Lower Laurentian.

"The so-called 'hypersthene rocks' of Dr. Emmons consist of labradorite and pyroxene or labradorite with hypersthene and some pyroxene, and hence are often designated as a Labrador series. In an article on the 'Laurentian Magnetic Iron-Ore Deposits in Northern New York," Charles E. Hall has grouped the magnetites in three series or horizons; the lowest, the Laurentian magnetites; second, the Laurentian sulphurous ores; and highest, the Labrador group, with its titaniferous ores."

Prof. George W. Maynard, in an elaborate paper on "The Iron-Ores of Lake Champlain," published in the *Journal* of the British Iron and Steel Institute, vol. i., 1874, p. 114, says that "mineralog-

* *Trans.*, xvii., 715.

† *Bulletin* of the New York State Museum, June, 1889.

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ically the ores of the Lake Champlain district may be considered under the following heads: The unaltered magnetites; the peroxidized magnetites (martites); the titaniferous magnetites. . . . The largest occurrence of the unaltered and titaniferous magnetites is in Essex county, the peroxidized ores being for the most part confined to Clinton county."

Prof. Smock states that "magnetite is one of the common minerals in the Adirondacks, and is widely distributed, both as a constituent or accessory mineral in rocks, and in beds of workable extent."

The tenth United States census reported 731,992 tons from all the mines in the Adirondack region.*

At present the producing mines of the Adirondack region may be grouped in three districts, those of Essex, Clinton, and Warren counties, respectively. The deposit to which this paper refers is in Essex county; and the mines in this county may be properly subdivided into two groups, viz., the "Port Henry" and the "Crown Point." The names indicate the towns on Lake Champlain from which the product of the mines is shipped. In both cases, the ore is obtained away from, and at considerable elevation above, Lake Champlain, railroads connecting the mines with the Lake ports.

Five miles northwest of, and 1200 feet above, Lake Champlain at Port Henry, in the town of Moriah, Essex county, is the village of Mineville, where are located the operations of the Port Henry Iron-Ore Co., and those of Witherbees, Sherman & Co., recognized in the trade as the "Port Henry mines." At present merely a general sketch of these remarkable deposits will be given, but at a future meeting a more complete paper will be presented upon the Adirondack magnetites.

The Port Henry Mines.

The group of iron-ore mines in the town of Moriah, Essex county, N. Y., which embraces the largest producers in the Lake Champlain district, is reached by the railroad connecting Mineville with Port Henry. To the latter point the ore mined is carried by cars over a road of standard gauge, having three switchbacks, as the difference of elevation requires a maximum grade of 1 to 20.75 (254.5 feet per mile). But as the loaded cars are carried from the mines to the lake the objection to heavy grades is less than if the conditions were reversed. The properties of the two companies operating these mines are contiguous, and to a stranger, the locality of some of the workings may

* This includes other mines than those mentioned in the Lake Champlain group.

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appear confusing, owing to their being so close together. Several of the workings of the two companies are connected.

The mines produce two distinct varieties of ore, one being a magnetite, known commercially as "Old Bed 21" ore, carrying apatite in such quantities as to raise the phosphorus beyond the Bessemer limit, and the other a magnetite carrying a siliceous gangue, but generally low in phosphorus, known as "New Bed" ore. These designations apply to ore from various openings, the name "Old Bed" being derived from the original opening made in 1824, and now owned by Witherbees, Sherman & Co., from which a large amount of ore has been won in workings extending 1100 or 1200 feet on the slope to a vertical depth of 300 feet. The No. 21 opening of the Port Henry Iron-Ore Company (which supplies the second portion of the commercial name for the ore), with its several shafts 200 feet south of the "Old Bed" opening, which was started in 1829, is however, the largest producer, and one of the most impressive mines to view in the world. The massive pillars of ore supporting the roof (also of ore), capped with rock to within 50 feet of the surface are exposed in the great open pit to a height of 200 feet, while the chambering extends much beyond the face and to a depth of about 350 feet vertically from the surface, the distance along the slope being 700 feet.

This mine (viz. No. 21) was originally operated on the outcrop, and Prof. Maynard says of it:*

"In 1853, the American Mineral Company was formed under the superintendence of Prof. Emmons, the State geologist, and they took a lease of the remaining property for the purpose of extracting the phosphates from the iron-ore to be used as fertilizers. . . . It seems almost incredible, but is none the less true, that this company, by their contract, agreed to turn over to the owners of the property all the iron-ore after they had extracted the phosphates from it, in addition to paying them a handsome royalty. As a natural consequence the company soon became insolvent."

The amount of apatite was either much greater on the outcrop than it is now, or the operators had abounding faith in the prospects of its utilization.

Northwest of the Old Bed opening is the Miller pit of Witherbees, Sherman & Co. This opening is on nearly the same level as the Old Bed and No. 21 ore-bodies, but the slopes are not the same, the ore strikes at different elevations, but in nearly parallel directions, in each of the three veins, and a fourth ore-body apparently lies

* *Journal* of the British Iron and Steel Institute, vol. i., 1874, p. 121.

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between the Old Bed and the Miller pit seams. An opening of the Port Henry Iron-Ore Company appears to be a continuation of the original Old Bed pit, while two openings of Witherbees, Sherman & Co. seem to be on the same ore-body as No. 21.

The possibility of these apparently distinct ore-bodies with dips varying from 30° to 70°, finally meeting in one great deposit, is claimed by some who have investigated the properties.

All of the above openings produce what is known commercially as "Old Bed 21" ore, but the percentages of phosphorus vary in the different workings, none, however, being Bessemer ore. The deepest workings are now nearly 600 feet vertically below the surface. The average shipping-analyses of the ore vary from 60 to 68 per cent. of iron, and from 0.5 to 2 per cent of phosphorus.

One and a half miles northeast of the openings described are two shafts producing similar ore. These are, a slope about 1200 feet and a vertical shaft about 800 feet deep, reaching at a depth of 100 feet from the surface a large body of ore. Neither of these is worked at present, as the railroad tracks do not reach them. Other exploitations have been made in the vicinity of the mines above described, and ore from various openings was taken years ago to blast-furnaces or forges which existed in the vicinity.

The openings producing "New Bed Ore" cover a greater territorial area on account of being made upon a vein which is apparently continuous, but divided into two or three layers or seams, occasionally cut by horses, while the various deposits of "Old Bed Ore" seem to be a series of lenses lying in *echelon*, or nearly parallel, of varying thickness and dip.

In Volume XV. of the Tenth Census of the United States, pages 106 to 112, is found a description, accompanied by maps and partial analyses, of the ore-bodies in the vicinity of Mineville.

The New Bed Mines.

Mr. W. H. Tefft, Superintendent for Witherbees, Sherman & Co., states that the layers of the New Bed vein average about 8 feet in thickness, increasing at some places to 20 feet, while the maximum thickness of the Old Bed ore-bodies is yet undetermined; thus, the Miller pit ranges from 10 to 100 feet, the Old Bed pit from 20 to 125 feet, while in No. 21 the pillars rest on ore and support a roof of ore, the body being known to exceed 380 feet in thickness. The newer shafts of Witherbees, Sherman & Co., the Bonanza and Joker, vary from 30 to 150 feet in thickness. The

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pitch of both New and Old Bed veins averages about 30 degrees from a horizontal line, but varies greatly, sloping to nearly vertical or flattening considerably, and sometimes returning in a roll.

The original New Bed opening has been continued along the strike for 2000 feet, reaching a vertical depth of 500 feet below the surface, and from the end of the present workings to the furthest exploitation of the vein is about 2½ miles. In this distance there are a number of workings operated by Witherbees, Sherman & Co.*

These openings, in order from south to north, are known as Roe Shaft, Little Pit, Big Pit, Walder's Shaft, Wasson Pit, Old Pit, Arch Opening, Lover's Hole, South Pit, North Pit, and Orchard Pit, the last six being on Barton Hill; then come the Fisher Hill openings, which are worked by the Port Henry Iron-Ore Company, which has, up to the present, followed the dip for 1000 feet. On lot 75, a mile north of Fisher Hill, Witherbees, Sherman & Co., are now reopening an old working, and recent explorations, made by Captain George S. Roe, of the Port Henry Iron-Ore Company, have also demonstrated a continuation of this ore-body. There is, therefore, no apparent prospect of speedy exhaustion of either the Old Bed 21 or the New Bed ore-deposits; and it is doubtful if there is a deposit in the United States which produces purer iron-ore than some portions of the New Bed vein.

Occasional horses of rock cut into the Old Bed lenses, while the New Bed has a series of faults caused by pyritiferous material, but sulphur is not found in the ore.

Mr. Charles E. Hall, in his monograph upon the "Laurentian Magnetic Iron-Ore Deposits of Northern New York," places the New Bed and Old Bed deposits in different geological horizons. The general strike of the ore-bodies is apparently north to northeast.

In the New Bed ore, at the Big Pit, three ore-bodies were found separated from one another by from 6 to 15 feet of rock-floor. The openings further up the slope of Barton Hill have a strike nearly north, with a southwestern dip.

The occurrence of two ores so close together, similar in the fact of carrying high percentages of iron, dissimilar in the phosphorus contents, and in the thickness or width and other characteristics of the veins, is interesting. The "Old Bed 21" workings are by far more impressive than the "New Bed," but either deposit is entitled to enthusiasm in description.

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* *Journal* United States Association of Charcoal Iron Workers, vol. viii., p. 218.

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From the New Bed deposits two commercial grades of ore are obtained, viz.: "New Bed pure," carrying from 60 to 68 per cent. of iron, and "New Bed lean," analyzing from 50 to 60 per cent. of iron. The proportions of these two grades, for a term of years, have been nearly equal, but lately, the "New Bed pure" ore has predominated, and out of about 150,000 tons mined by Witherbees, Sherman & Co., in 1889, nearly 100,000 tons were of "New Bed pure" grade.

In mining this New Bed ore, occasional pockets of remarkable purity are found, and from some of these excellent crystals of magnetite are obtained. The mass of the ore is crystalline, and while perfect or modified octahedral crystals are frequently obtained, there have been but few instances where these crystals are found in great abundance. The pockets of abnormally rich ore, and the occurrence of a great number of crystalline forms have been noted by Mr. Tefft in nearly all of the New Bed openings, but only three or four of these have been of sufficient magnitude to excite attention, and the pocket or lense described in this paper has by far exceeded any other find in extent. Its discovery is renewed evidence of the value of prospecting in workings before abandonment. The workings are shown in plan and section in Figs. 1 and 2. Operations had been prosecuted on the Lover's Hole ore-body until it was deemed advisable to abandon the slope in use and make a new opening; before doing so, however, exploration holes were bored, and one of these after passing through 18 feet of floor showed rich ore below. A shaft was sunk, and this ore was found to be remarkably high in iron, as will be seen from the analyses given on pages 758 and 759.

The Lover's Hole Mine.

Fig. 1 is a plan, and Fig. 2 presents two vertical sections, of the workings in this mine.

Exploitations had been prosecuted upon an upper ore-body, using the main adit or incline shown by parallel lines on the accompanying plan, in the directions X to Y and Y to Z, and it was in the vicinity of the point C that a drill-hole from the upper workings penetrated a large body of ore below, which has since been worked, and in which the crystals were found. The general relation of the upper and the lower ore-bodies to each other and to a horizontal line is shown by the two sections; but the position of the two ore-bodies, as exploited, will not exhibit the exact section as shown, for, with the exception of the portion in the vicinity of C, the two workings

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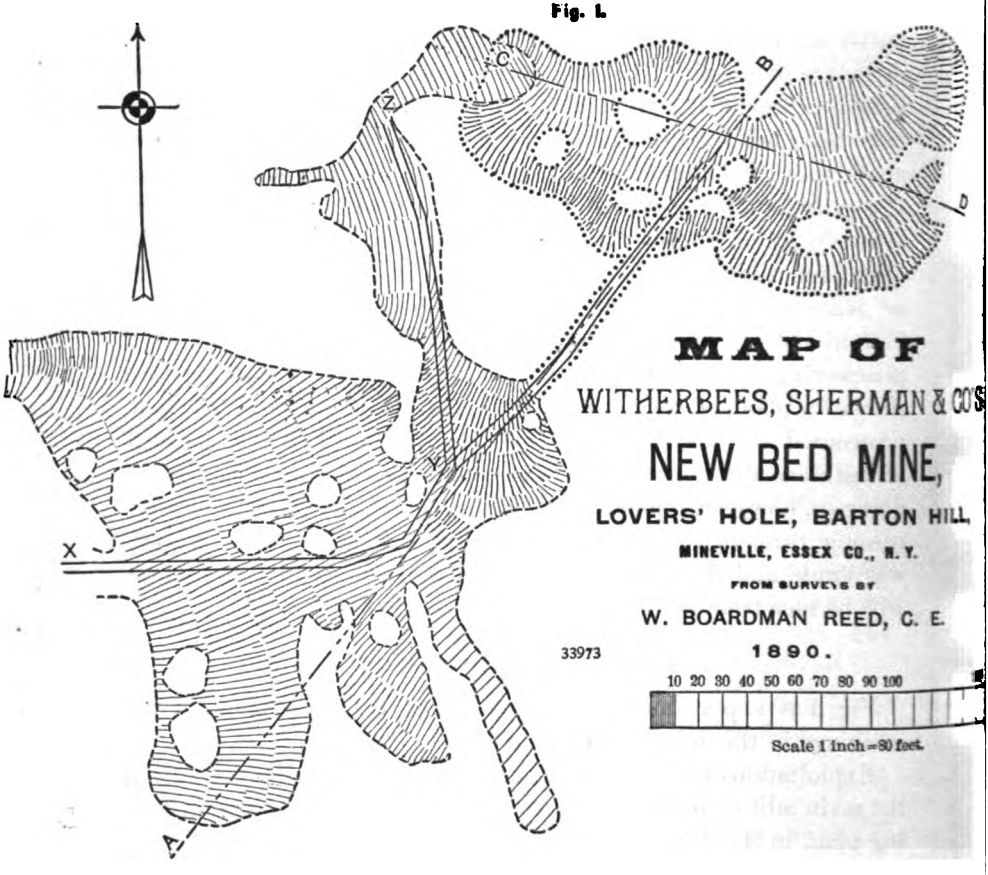
were not over each other when the map was made. The section may, however, be considered as a true one (except in the fact of indicating the pillars) along the line AYB, for the lower workings. The upper portion, namely, that between Y and Z, shows the relative position of the upper workings, with regard to its slope and distance, above the lower. The section AB is longitudinal on the slope; the section CD is transverse, taken at the then lower portion

Fig. 1.



of the workings, and exhibits the supposed, rather than the known, position of the upper ore-body relative to the lower one. In the plan, Mr. Reed has indicated by topographical hatching the approximate roof contour of the mine. In the sections, the hatching by parallel lines is intended merely to indicate the rocks inclosing the ore; and the vertical position of these sections on the paper is approximately

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true to the bottom of the page as a horizontal line, thus indicating the general slope of the exploitation.

XYZ, Fig. 1, shows the direction of the workings in the upper level or ore-seam, and the hatched lines adjacent to XYZ, and embraced by a broken line contour, indicate the general plan and underground topography of the upper workings from which the ore had been removed except at the pillars, which are left blank. After the test-hole at C had developed the rich ore at the lower level, the incline was continued from Y towards B, the area shown by hatched

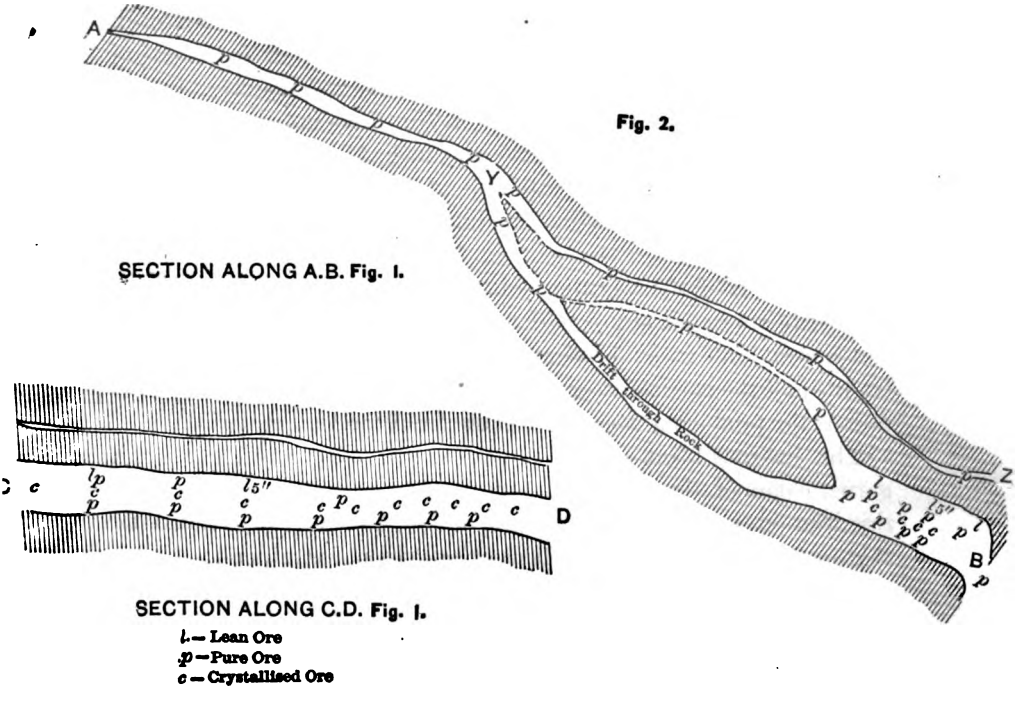

Fig. 2.

lines embraced in a dotted contour exploited and the ore removed, except where pillars are shown. In connecting the upper and lower workings, the slope or adit passed through a body of about 120 feet of barren rock. The longitudinal section along the line, AB, Fig. 2, shows the general direction of the drift, and also the relative position of the upper and lower ore-body, the upper bodies extending from Y to Z, and the lower body passing over the horse of rock, but thickening very much beyond it. The approximate position of the upper and lower ore-bodies is also shown on a transverse section on the line CD, Fig. 2. In each of the

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sections the letters, *cc*, indicate the approximate position of the greater number of octahedral crystals. The letters, *pp*, represent the rich ore in which the crystals are not so well defined, and the letters, *ll*, show the occasional occurrence of lean ore close to the walls.

Mr. W. Boardman Reed, mining engineer, who made the sketch-map of these operations from his elaborate surveys of the mines, found the lean ore as usual along the walls, but near the center of the mass of pure ore, in a space 20 feet wide and 10 to 12 feet thick, most of the crystals were obtained, although some of these and numerous so-called cleavage masses were found scattered throughout the ore-body. Some remarkably unique specimens were obtained, in which individual crystals of considerable size occur in a mass of smaller but less perfectly formed crystals acting as a matrix. One of these shows a mass of fine crystalline ore, in which is embedded a perfect octahedron with faces of over one inch, resting loosely in the socket, from which it is readily lifted. Among the crystals are many with truncated edges, and others modified by the unusual development of parallel faces.

In portions of the workings where the ore is lean, the crystals of magnetite are found mixed with quartz (white, and also with a hematite stain) and with chlorite (ripidolite), but the mass crumbles readily and the particles separate easily, offering very satisfactory conditions for producing concentrated ore from the lean material.

The lustrous black of the magnetite, the clear white, pink and garnet colors of the quartz, and the green tint of the chlorite gives this lean ore a brilliant appearance, which, however, is excelled by that of the portions of the ore-body where the purer mineral is found, and where innumerable crystal faces reflect the rays from the electric lights. A visit to the operations of the Lover's Hole will satisfy any one viewing it by electric light that the miners have apparently good reason for adding to their mining vocabulary the word "diamonds" to designate the octahedrons which abound.

Since the map was made, and the above written, operations have been continued, and at a late visit (April, 1890) a large nest of octahedral crystals was exposed in an approximate continuation of the line YB, where the hanging-wall had rolled to nearly a vertical dip. A further extension of the exploitation of the line CD has connected, beyond D, the workings of Lover's Hole and South Pit, the connection being made through a mass of rich ore, exhibiting the pseudo-cleavage and smooth faces. The evidence that these two operations are upon the same ore-body, demonstrates that the abnormal extent of the crystalline ore is merely a local enrichment of the vein.

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Through the courtesy of Mr. Uno Sebenius, assistant superintendent for Witherbees, Sherman & Co., I have been enabled to present to the United States Geological Survey an interesting suite of the minerals and rocks found in this opening, which shows that the rock along the foot-wall, while resembling in appearance a lean magnetite, is practically a close intermixture, in which hornblende predominates, and this rock does not differ materially from that along the hanging wall. This hanging-wall rock, however, shows a coating in which hornblende, apatite and other minerals occur. All of the minerals have not been absolutely determined, but among them may be mentioned hornblende, quartz, feldspar, apatite in crystals, sphene, chlorite (ripidolite).

Mr. Sebenius, in referring to the geological features of the Lover's Hole deposit, considers it to follow very regularly the dip of the strata of the gneissic rock in which it occurs, and claims that in no place does the vein cut the strata, thus apparently contradicting the theory that the vein is erupted. From the top of Barton hill to the bottom, and throughout the mines, this granulitic gneiss (a rock of sedimentary origin) has the same general appearance.

In this rock he says "the ore has been deposited during the azoic age in the same manner as to-day it is being formed in sand banks elsewhere, the *débris* of magnetite and other minerals accumulating and separating under the influence of water and pressure." From this he argues that the ore is of practically the same age as the enclosing rock. He claims that this separation of the heavier from the lighter is "probably the reason why hornblende, epidote, augite, etc, are always found together with the ore, and that the heavy strata of hornblende are to be found in the iron-bearing rocks and in the vicinity of magnetic veins and beds of hematite."

In regard to Lover's Hole and other New Bed workings, Mr. Sebenius claims that after the deposit of stratified rock and ore was completed, a tilting action took place, which has given a wavy contour, thus accounting for the variation in dip.

Chemical Composition.

The output of nearly, if not quite, 40,000 gross tons of ore taken, up to the close of the year 1889, from this Lover's Hole opening, shows the average iron contents to be 68.6 per cent. and the average phosphorus contents to be 0.033 per cent. These averages represent *all of the ore* removed from this working, and are obtained, not from

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selected samples, but by adding together the thirty analyses of the ore made between March 2d and December 28th, 1889, and dividing by thirty. These determinations were made of samples taken practically from each car-load shipped by rail from the mines to Port Henry, and delivered to the laboratory every week or ten days when the sampling receptacle was filled.

The thirty determinations are given below to indicate the extreme range of composition, and they are arranged for examination in the order of their iron contents in preference to using the chronological order adopted in the laboratory by Mr. T. Reed Woodbridge, chemist for Witherbees, Sherman & Co., at Port Henry, N. Y., who made all of the analyses, and also furnished other valuable data for this paper.

Concerning the determinations, the highest percentage of iron, viz., 72 per cent., found in the early development of this working, was obtained from a sample in which crystal forms were abundant, while the lowest iron contents, viz., 61.2 per cent., was evidently determined from an excess of lean ore or the presence of wall-rock after the workings had been considerably extended, for we find the next lower analysis shows 65.10 per cent. of iron. The laboratory dates of these two analyses indicate this to be the case. The one determination showing 72 per cent. of iron was checked by a second analysis; there are also two analyses between 71 and 72 per cent., and seven between 70 and 71 per cent. One-third of the determinations, therefore, indicate 70 per cent. or over of iron.

A personal examination of the sample from which Mr. Woodbridge obtained by duplicate analyses the remarkable yield of 72 per cent., shows but few unmodified octahedral crystals, the predominating forms being the massive striated ore with pseudo-cleavage, which, upon fracture, shows a denser structure than is generally exhibited by the more perfect octahedrons.

The variations in the phosphorus contents are from 0.011 to 0.060, but the two determinations in which the phosphorus reached the latter figure showed in the same samples 67.20 and 68.45 per cent. of iron, so that in no instance do analyses show that the phosphorus exceeds 1 part in 1120 of iron.

The following are the thirty determinations above referred to, kindly supplied by Mr. Woodbridge:

	Per cent. of Iron.	Per cent. of Phosphorus.
1,*	72.00	0.018
2,	71.70	0.022
3,	71.20	0.026

* Two determinations.

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	Per cent. of Iron.	Per cent. of Phosphorus.
4,	70.60	0.023
5,	70.40	0.011
6,	70.40	0.022
7,	70.40	0.048
8,	70.25	0.045
9,	70.20	0.021
10,	70.20	0.051
11,	69.90	0.025
12,	69.90	0.046
13,	69.50	0.013
14,	69.20	0.020
15,	69.10	0.023
16,	69.10	0.031
17,	69.00	0.025
18,	68.70	0.030
19,	68.45	0.060
20,	67.80	0.038
21,	67.70	0.054
22,	67.20	0.035
23,	67.20	0.060
24,	67.00	0.042
25,	66.70	0.030
26,	66.60	0.026
27,	66.10	0.036
28,	65.20	0.042
29,	65.10	0.048
30,	61.20	0.024
Average,	68.60	0.033
Maximum,	72.00	0.060
Minimum,	61.20	0 011

Mr. Woodbridge also made careful analyses of the octahedral crystals with the following results, as reported by him.

"Five crystals, weighing about 80 grammes, were crushed and passed through an 80-mesh sieve. The ore was not crushed fine or rubbed in an agate mortar, but weighed from 80-mesh sieve to avoid loss and contamination with metallic iron."

"The specific gravity of the powder was 5.119. Ten grammes heated one and one-half hours at 120° C. lost 0.001 gramme or .01 per cent. The analyses are given below:

Magnetic oxide of iron (Fe_3O_4), average of several determinations,		98.850	
Phosphoric acid (P_2O_5), average of several determinations,		.009	
Silica (SiO_2),	.110		125
Alumina (Al_2O_3),	.240		.240
Lime (CaO),	.280		.310
Magnesia (MgO),	.350		.330
Titanic acid (TiO_2),	.147		.150
Sulphur (S),		none	
Manganese (Mn),		trace	
Total,	9.969		100.014

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Average of several determinations: iron, 71.58; phosphorus, .004."

"Two analyses were made to determine iron existing as FeO, but the results, 22.60 and 23.60, differed so materially, that two more determinations were made in samples weighing 0.9525 gramme and 1.2310 grammes, taking greater precautions to prevent oxidation of the solution, the results, 23.727 per cent., and 23.753 per cent. giving an average of 23.74 per cent., being very near the theoretical value for FeO in Fe_3O_4."

"The first determination of TiO_2 (.147) was made with a 5-gramme sample precipitating as acetate from the hydrochloric acid solution of the ore and subsequently purifying by fusion in alkaline carbonates and precipitating from sulphuric acid solution in presence of sulphurous acid. The second determination (.150) was made by the ordinary method of fusion of 1 gramme with potassium bisulphate, the first precipitate of TiO_2 being purified by fusion as above."

The crystals which are found in the Lover's Hole present many modifications, although they are all in general form octahedral. A collection of them was presented to Prof. Geo. A Koenig, PhD., of the Department of Metallurgy and Mining in the University of Pennsylvania, who has very kindly supplied the following information concerning them:

"The magnetite crystals of this ore-body disclose several interesting crystallographic features. The predominating form is, as usual, octahedron (0) combined with the rhombic dodecahedron (∞ 0.) With these two forms occur very generally forms of the pentagonal hemihedry, not often observed in magnetite: the diplohedron $\left(\frac{m\,0\,n}{2}\right)p$ and the pentagonal dodecahedron $\left(\frac{m\,0\,\infty}{2}\right)p$. On three crystals the cube (∞ 0 ∞), and on two the icositetrahedron (m 0 m) have been noticed. The faces of these modifying forms are always dull and sometimes curved from the presence of vicinal faces. The most noticeable feature is the *partial* development of the modifying forms. I have one crystal before me with only *one* face of the diplohedron quite largely developed. Other crystals show one pair of diplohedral faces in one apex, the other solid angles appearing altogether unmodified. But there are some in which the diplohedron occurs on two apices, and one crystal (the largest of the lot being about 2½ inches) shows one face of a penta-

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gonal dodecahedron (probably $(\frac{2\,0\,\infty}{2})$), each in two of the six apices of the crystals."

"The *pseudocleavage* by which prismatic habitus of the octahedron is brought about is, in my estimation, a peculiar phenomenon of growth. The faces of such cleavable or cleft specimens show invariably a covering of ripidolite. There must have been temporary cessation in the growth of the magnetite, and during these stoppages the scales of ripidolite or prochlorite were crystallized. Then the magnetite resumed its crystallization, the mass action of the already existing crystals exerted its influence through the intervening film of ripidolite, and the molecules arranged themselves parallel to those of the preceding crystallization. Very fine specimens of this type have recently been found growing up in steps and gaps. The entire group can easily be recognized as the skeleton of a very large octahedron."

"As easily this group will pass into prismatic, tetrahedral or octahedral parts, always along a film of ripidolite, thus producing the impression of cleavability."

No attempt has been made to enter into an elaborate discussion of the occurrence, composition or crystal forms of this interesting ore-deposit. The purpose of this paper is rather to place on record the fact of the existence of this remarkable body of ore giving only such data concerning its location, the geological, mineralogical and chemical features as will make the paper of value. In view of what has been written, it is natural that the question should arise as to how the ore from the opening known as the "Lover's Hole" compares with that from other openings upon the "New Bed" vein, and to answer this we have been granted permission by Witherbees, Sherman & Co., to examine the laboratory-record covering the determinations of about 100,000 tons of New Bed pure ore from the Port Henry mines, made by Mr. Woodbridge in the year 1889. Below we give the number of analyses made from ore obtained from each opening in the New Bed vein, operated during the year 1889, the average of these analyses in iron and phosphorus, and also the maximum iron contents as determined in the laboratory, supplying a record which we believe is unique in the annals of magnetic iron ore mining.

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Analyses of New Bed Ores.

Name of Mine.	Number of Analyses.	Average Iron.	Average Phos.	Highest. Iron.	Phos.*
Lover's Hole,	30	68.60	0.033	72.00	0.018
North Pit,	26	62.99	0.024	69.70	0.029
South Pit,	28	63.48	0.047	68.90	0.036
Orchard Pit,	22	63.34	0.020	67.00	0.023
Roe Shaft,	4	63.40	0.045	64.70	0.061
Big Pit,	21	66.81	0.032	70.00	0.026
Little Pit,	15	65.65	0.027	68.60	0.022
Total and averages,	146	65.12	0.0289		

These show that the average iron contents of 100,000 tons of New Bed ore (65.12) coincides closely with the second low iron contents of Lover's Hole ore (65.10), but the average from two other workings only, viz., Big Pit and Little Pit, exceed this second low analysis. None of the averages, however, approach the lowest analysis of Lover's Hole ore (61.20). The richest ore from the various openings, as shown from the analysis, is below the average of Lover's Hole opening (68.60) in Orchard Pit and Roe Shaft only.

The phosphorus average of Lover's Hole ore (0.033) exceeds that of four of the openings and is greater than the average phosphorus of all the openings. Comparing these as to the amounts of phosphorus in one hundred parts of iron we have: average of Lover's Hole ore, .0481 per cent.; average of all New Bed pure ore, .0444 per cent.

* Phosphorus contents in this column are from the same samples in which the iron is determined.

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A SYSTEM OF RAIL-SECTIONS IN SERIES.

BY P. H. DUDLEY, NEW YORK CITY.

(Washington Meeting, February, 1890.)

A QUARTER of a century of service of steel rails on our oldest railroads, many of which have changed their standard sections three or four times, has furnished, and is furnishing, excellent opportunities to study their wear and behavior in the track. Such experience is valuable for the design of new sections, now required to meet the constantly increasing severity of railroad service.

Before presenting the new sections for consideration, I will briefly call attention to some features of past experience.

For convenience of study, rails may be separated according to their weight and height into four groups, viz.:

1. Rails weighing per yard from 52 to 60 pounds, and 3½ to 4 inches high.
2. " " " " " 60 " 72 " " 4⅛ to 4½ " "
3. " " " " " 72 " 90 " " 4¾ to 5 " "
4. " " " " " 85 or more " " over 5 " "

Experience in this country is confined mainly to sections in groups Nos. 1, 2 and 3.

During the past few years I have had the opportunity to run my car over many thousands of miles of railway laid with the principal sections in use, and to obtain diagrams showing their deflections, permanent set, and, in a great measure, their manner of wear.

The First Group.—The first steel rails imported, and those manufactured here, were light sections—included in the first group—and many of them, being rolled in rolls used for rolling iron sections, have pear-shaped heads, 2⅜ to 2½ inches wide, the base being heavy and from ¼ to ½ inch narrower than the height.

These rails were laid upon ties of solid timber, at least 8 inches face, spaced 2 feet from center to center. The moments of inertia of these rails were low, only ranging from 8 for the lightest sections to 12 for the heaviest. The deflections were, of course, great; but the quality of the metal was excellent. It was physically hard and

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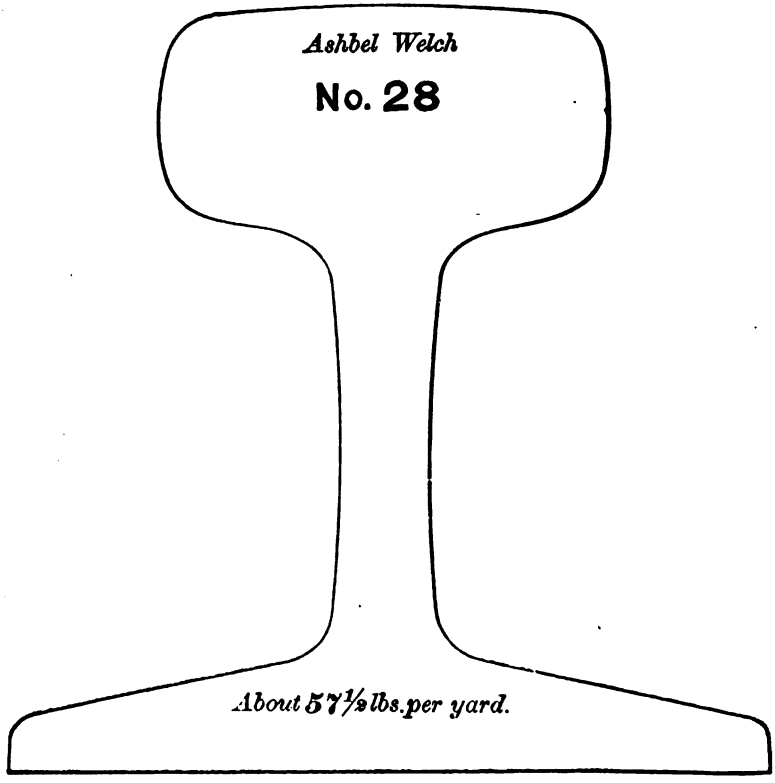
tough, did not easily take a permanent set and wore very smooth. The loss of metal per million tons of traffic was, for Sir John Brown & Co.'s rails, on tangents, .01 to .015 pound per yard; on gradient tangents, .015 to .02 pound per yard; on gradient curves, .025 to .03 pound per yard. The rails of Charles Cammell & Co., Landore & Siemens and the Barrow Company were quite as good. Several of the American brands were excellent, and, by many railroad officials, considered equal to the imported.

The wheel-tonnage, however, was much lighter than at present, being 4 to 6 tons per locomotive-driver; 2 to 2½ tons per freight-

FIG. 1.



car wheel; 3 to 3½ tons per passenger-coach wheel. The cutting of ties under these rails was rapid even for the light tonnage.

The Second Group.—Observing these features led to the design of stiffer and heavier sections, in Group No. 2 of the Ashbel Welch type, of 60, 63, 65 and 67 pounds per yard. His design was made

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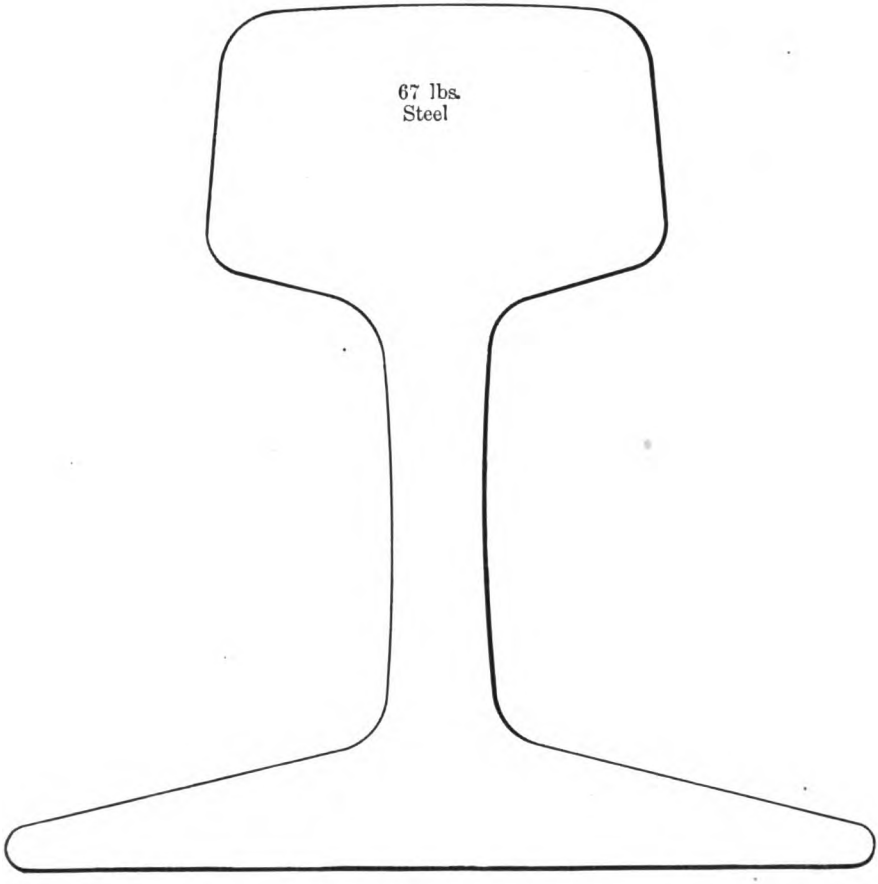
in 1866, and by 1870 several sections were in service. These rails were from $4\frac{1}{8}$ to $4\frac{1}{2}$ inches high, had heavy webs and bases, the latter usually $\frac{1}{4}$ to $\frac{3}{8}$ inch less in width than the height of the section. This type is shown by Fig. 1.

The metal in many of these sections was intended to be fully equal to that of the earliest rails; and it wore well, though, as we

FIG. 2.



now know, the loss of metal should be greater per million tons of traffic, than on the lighter and more flexible rails.

Many thousands of tons of these rails were laid by the trunk-lines, some of them being still in service, while a large part was removed, to be replaced with heavier sections, on account of increased traffic, and not because the rails were entirely worn out.

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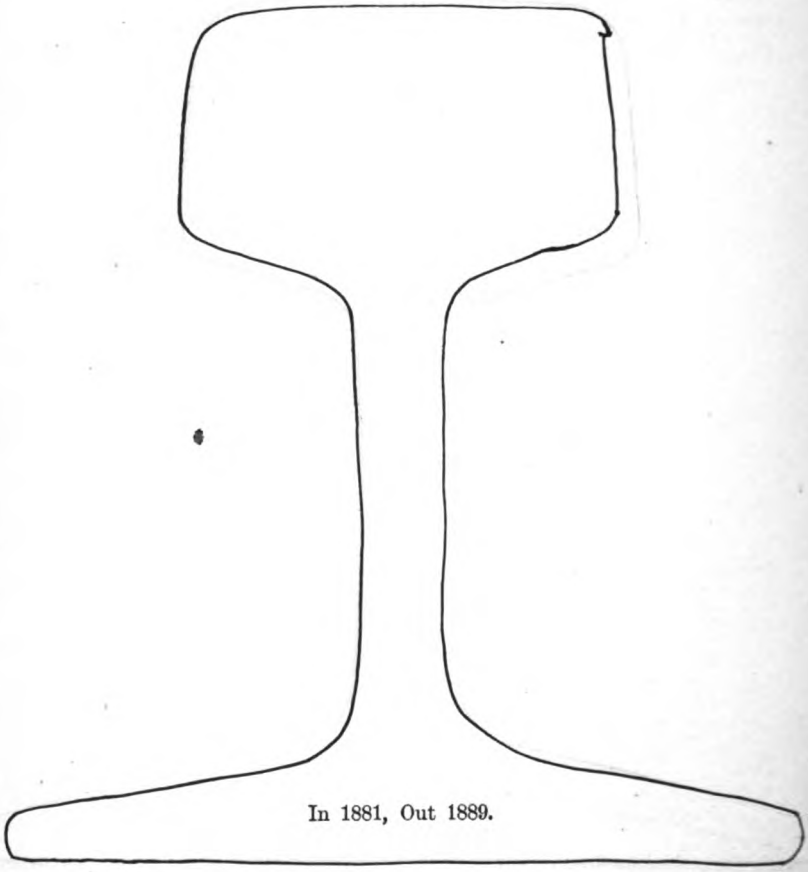
The satisfactory observed wear of these rails led to the conclusion that the only wear was upon the head, and, therefore, that the governing principles of new designs of rail-sections should be a shallow base as wide as the height, and a thin web supporting a narrow and deep head for wear. After 1873, these principles were adopted in practice, and millions of tons of rails of this type, represented

FIG. 3.



In 1881, Out 1889.

by Fig. 2, were put into the track (most of the sections being included in Group No. 2) prior to the year 1880.

In many cases, where the change was made from one type of section to the other, the weight per yard remained the same. These being put in the same tracks, carrying yearly the same tonnage, have furnished excellent opportunities to study the comparative wear of

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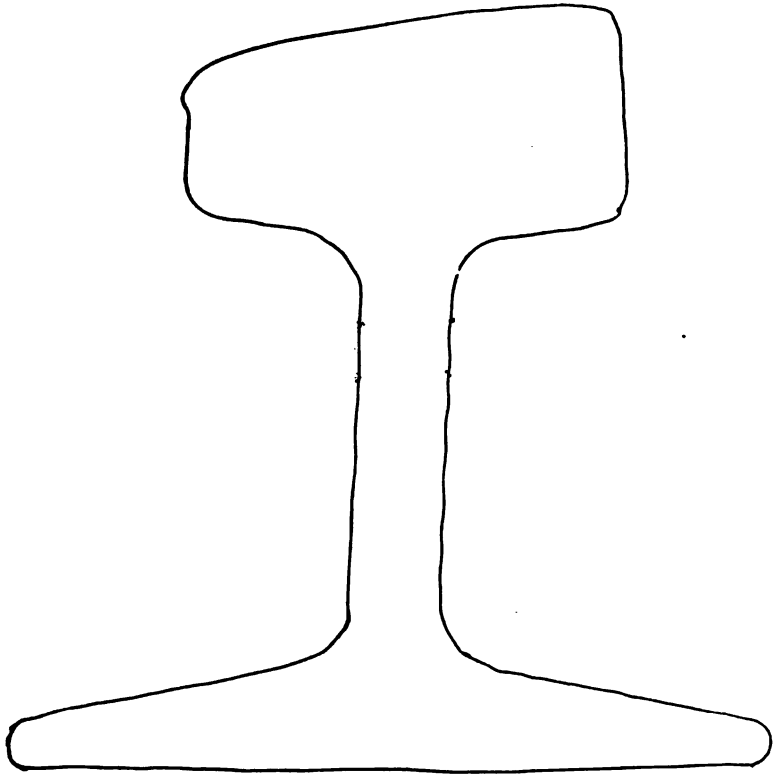
the two types. It is now well known that the wear of the deep-headed type has been far more rapid than that of the shallow-headed type. Rails of the former class take a permanent set in the track much more quickly, and many have failed by loss of metal on the base, through wear and oxidation (See Figs. 3 and 4).

The tendency of these deep-headed and light-based rails to take a permanent set on the track, is much greater than that of rails having

FIG. 4.



heavier bases, because, the neutral axis of the section being so high, the metal in the head is readily overstrained, taking a set in the head of the rail, which assumes either the first or second form, as shown by Figs. 10 and 11 in my previous paper (*Trans.*, xviii., 228).

After the rails have taken a permanent set, the receiving-ends of soft rails with narrow heads flatten out, making a depression in the surface of the rail which cannot be surfaced.

Last season, I had under observation several miles of 67-pound

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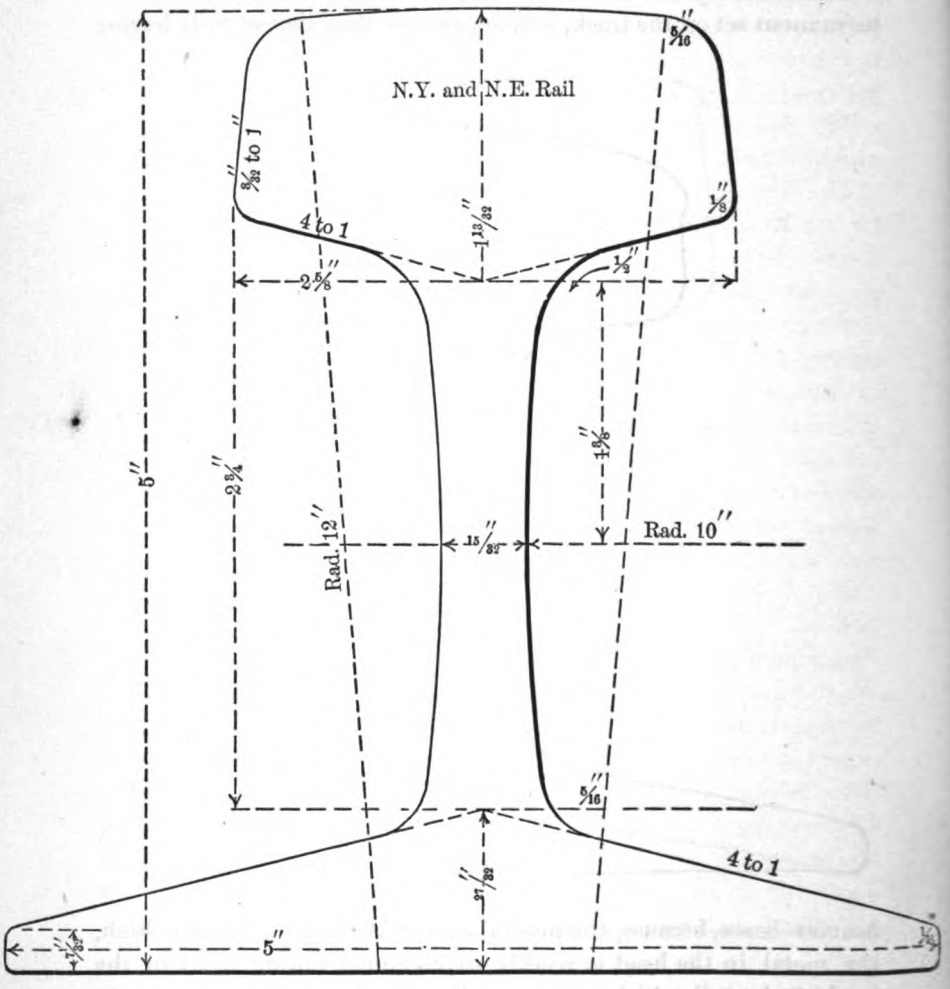
rails of 1882, width of head $2\frac{1}{4}$ inches, in which a number of the receiving-ends of the rails had widened to $2\frac{3}{4}$ inches. The rails were not only examples of deep-headed rails, but also of the theory that chemically soft rails are best for wear. The three 72-pound

FIG. 5.



sections in Group No. 2 are only alike in height and weight; one has a 5-inch base and Sayre type of head (a difficult section to make physically hard and smooth, though well executed by one manufacturer), and stands up well under the traffic.

Two other manufacturers, believing that soft rails were the best

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for wear, made them much softer, but these show an increase of wear compared with the harder ones, and some of them have taken so much permanent set as to require removal and re-straightening.

The Third Group.—Rails, 4¾ to 5 inches high, represent recent sections designed to meet the exigencies of present railroad service.

There is one 72-pound section, 4¾ inches high, base same width, large radii of corners on top of head, the sides of the latter inclining to the web of the rail. This is the "Forney type," and is used by one road.

The Sayre section of a 76-pound rail, 4¾ inches high, 4¾ inches width of base, is used by a few roads.

The first of the 5-inch rails, the 80-pound rail which I designed for the New York Central and Hudson River railroad in 1883, is described in my previous paper (page 228), and, I may add, is the pioneer of the broad and shallow heads and thick bases for heavy rails.

With an increase of 8 pounds over the 72-pound 4½ inch rails, a gain of 40 to 50 per cent. in stiffness is made, which is an important advantage in saving of ties and of labor in maintenance of track, and in producing a marked reduction in train-resistance.

Nearly six years' experience with some of these rails (about 200 miles of track laid with them; 30,000 tons to be put in this year) is good evidence that the section is meeting the expectations of the consumer.

Fig. 5 shows a section I designed in July, 1888, for 75-pound, 5-inch rails, 5-inch base, for soft-wood ties for the New York and New England railroad, then under the general management of Mr. W. P. Shinn, ex-President of this Institute, and now President of the American Society of Civil Engineers. Mr. Shinn has had large experience, not only as a consumer, but as a manufacturer of rails. After making an inspection of his road, I suggested that, by using 75 pounds of metal for their rails, a section could be designed nearly 50 per cent. stiffer than their present 66-pound rail, which would be of great advantage for the heavy wheel-tonnage of their coal-cars. Below is a table, giving the deflections of this rail under loads, upon supports 30 inches apart, which shows that the section is a very stiff one, as I intended it should be, for their service, with a minimum amount of metal. (See tests of No. 6700.)

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ORDNANCE DEPARTMENT, UNITED STATES ARMY.

REPORT OF MECHANICAL TESTS.

[Made with the United States Testing-Machine, capacity 800,000 pounds, at Watertown Arsenal, Mass., May 14, 1890, for P. H. Dudley, C. E., New York, N. Y.].

TRANSVERSE TESTS AND TESTS BY TENSION.—STEEL RAILS.

Description of Rails.

No. 8.—80-pound steel rail, new piece, received from P. H. Dudley, June, 1888. Branded "A. R. I. and S. Co., Troy Steel, 1884, 23."

No. 23.—75-pound steel rail, D. and L. S., 1889. Branded "Lackawanna I. and C. Co., Scranton, 4–89." Forwarded by P. H. Dudley from New York and New England Railroad. (His No. 6).

No. 33.—Piece of new rail. Branded "Scranton Steel, 3–89."

No. 6700, Mark 23.—Steel rail, from New York and New England Railroad, 75 pounds. Branded "Lackawanna I. and C. Co., Scranton, 4–89." Rail supported on end-bearings, 30″ apart. Loaded at the middle.

Applied Loads. Total lbs.	Deflections.	Successive Deflections.	Deflection Sets.	REMARKS.
2,000	.0	.0	.0	Initial load.
4,000	.0025	.0025	...	
6,000	.0051	.0026	...	
8,000	.0075	.0024	...	
10,000	.0098	.0023	.0004	
12,000	.0120	.0022	...	
14,000	.0142	.0022	...	
16,000	.0167	.0025	...	
18,000	.0190	.0023	...	
20,000	.0211	.0021	.0007	
22,000	.0235	.0024	...	
24,000	.0258	.0023	...	
26,000	.0282	.0024	...	
28,000	.0302	.0020	...	
30,000	.0322	.0020	.0008	
32,000	.0344	.0022	...	
34,000	.0367	.0023	...	
36,000	.0388	.0021	...	
38,000	.0409	.0021	...	
40,000	.0430	.0021	.0009	
42,000	.0452	.0022	...	

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Applied Loads. Total lbs.	Deflections.	Successive Deflections.	Deflection Sets.	Remarks.
44,000	.0473	.0021	...	
46,000	.0494	.0021	...	
48,000	.0513	.0019	...	
50,000	.0534	.0021	.0011	
52,000	.0552	.0018	...	
54,000	.0573	.0021	...	
56,000	.0592	.0019	...	
58,000	.0614	.0022	...	
60,000	.0633	.0019	.0012	
62,000	.0660	.0027	....	
64,000	.0676	.0016	...	
66,000	.0696	.0020	....	
68,000	.0718	.0022	...	
70,000	.0738	.0020	.0015	
72,000	.0759	.0021	...	
74,000	.0779	.0020	...	
76,000	.0800	.0021	...	
78,000	.0820	.0020	...	
80,000	.0843	.0023	.0020	
82,000	.0865	.0022	...	
84,000	.0886	.0021	....	
86,000	.0910	.0024	...	
88,000	.0933	.0023	...	
90,000	.0960	.0027	.0033	
92,000	.0987	.0027	...	
94,000	.1011	.0024	...	Scale starts off head in vicinity of middle bearing.
96,000	.1038	.0027	...	
98,000	.1070	.0032	...	
100,000	.1105	.0035	.0075	
102,000	.1142	.0037	...	
104,000	.1179	.0037	...	
106,000	.1241	.0062	...	
108,000	.1313	.0072	...	
110,000	.1400	.0087	.0260	
112,000	.1508	.0108	...	
114,000	.1590	.0082	...	
116,000	.1707	.0117	...	
118,000	.1869	.0162	...	
120,000	.2025	.0156	.0760	
122,000	.2185	.0160	....	
124,000	.2315	.0130	...	
126,000	.2509	.0194	....	
128,000	.2686	.0177	...	
130,000	.2900	.0214	.1502	Transverse test discontinued.

No. 6701, Mark 33.—New York Central and Hudson River Railroad. 80-pound rail. Actual weight of piece tested, 78 pounds per yard. P. H. Dudley. New rail, branded "Scranton Steel Co., 3–89." Rail supported on end-bearings, 30 inches apart. Loaded at the middle.

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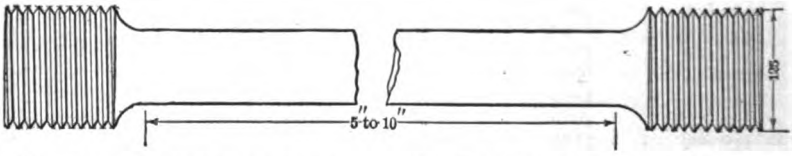
Applied Loads. Total lbs.	Deflections.	Successive Deflections.	Deflection Sets.	REMARKS.
2,000	.0	.0	.0	Initial load.
10,000	.0095	.0095	.0	
20,000	.0205	.0110	.0002	
30,000	.0316	.0111	.0002	
40,000	.0423	.0107	.0003	
50,000	.0525	.0102	.0003	
60,000	.0623	.0098	.0004	
70,000	.0723	.0100	.0005	
80,000	.0823	.0100	.0008	
90,000	.0927	.0104	.0015	
92,000	.0949	.0022	...	
94,000	.0969	.0020	...	
96,000	.0990	.0021	...	
98,000	.1015	.0025	...	
100,000	.1035	.0020	.0027	
102,000	.1061	.0026	...	
104,000	.1084	.0023	...	
106,000	.1108	.0024	...	
108,000	.1133	.0025	...	
110,000	.1160	.0027	.0052	
112,000	.1189	.0029	...	
114,000	.1215	.0026	...	
116,000	.1243	.0028	...	
118,000	.1277	.0034	...	
120,000	.1304	.0027	.0095	
122,000	.1349	.0045	...	
124,000	.1383	.0034	...	
126,000	.1423	.0040	...	
128,000	.1478	.0055	...	
130,000	.1545	.0067	.0228	
132,000	.1640	.0095	...	
134,000	.1721	.0081	...	
136,000	.1820	.0099	...	
138,000	.1947	.0127	...	
140,000	.2100	.0153	.0654	Transverse test discontinued.



TENSILE SPECIMENS, TAKEN FROM HEADS OF STEEL RAILS.

Test Number.	Mark on Specimen.	Dimens's in inches. Diam.	Sectional Area. □".	Elastic Limit. Total lbs.	Elastic Limit. Lbs. per □".	Ultimate Strength. Total lbs.	Ultimate Strength. Lbs. per □".	Elongation in 5 and 10 Inches. In's.	Elongation in 5 and 10 Inches. Pr. ct.	Area at Fracture.	Contraction of Area. Pr. ct.	Appearance of Fracture.
6702	8	.798	.50	22,840	45,680	43,210	86.420	1.04	20.8	Diam. .59=□" .273	45.4	Silky.
6703	23	.693	.377	20,730	55,000	45,520	120,740	1.26	12.6	" .59= .273	27.6	Gr., silky center.
6704	33	.695	.379	23,650	62,400	47,170	124,460	1.20	12.0	" .61= .292	23.0	Gr., silky cen. spt.

Correct.—J. E. HOWARD.

D. W. FLAGLER, *Lieut. Col. Ordnance Dept., U. S. A., Commanding*

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In 1889, 1800 tons of this type were put into service, making excellent tracks, fully as smooth as those made by heavier 5-inch rails. It represents a type of section, very stiff, but of light weight, which can be economically used in many localities.*

There are several sections of 85-pound 5-inch rails of this group, all of the deep-headed type, with the exception of the Delano section. There is one 90-pound rail, 5 inches high, same width of base, of the deep-headed type, though the base is heavy and web $\frac{7}{16}$ of an inch thick. For the weight of metal, the moment of inertia of the section is low.

The Fourth Group.—Of the fourth group, but a few rails are in the track in this country, while abroad there are a large number. These stiff sections will be required here, as soon as fewer ties are used per mile, or upon the general substitution of metal ties for wooden ones, with greater spacing. The New York Central company has put in a few hundred metal ties, spaced 3 feet from center to center, under the 80-pound rails.

Of Groups Nos. 1, 2 and 3, each succeeding one has shown a decided reduction in the deflection of the rails, and consequent improvement in the smoothness of the tracks under the trains, provided the rails were well finished.

Physical Hardness of Old-Style Rails.

Whether the scale of hardness of the older rails is estimated by indentation with the pyramidal cutter, elastic limits of the steel, or ultimate strength, they are found to be much harder than most of the rails of the deep-headed type.

Samples of rails of one brand of manufacture, from their first yearly output to date, for experiments upon hardness, are difficult to obtain. Those which I have succeeded in getting, indicate a considerable reduction in hardness from 1870 to 1885, and an increase since the latter date. This is shown in the curve in Fig. 6, in which the vertical lines represent the year of manufacture, and the horizontal lines the scale of hardness. The wear of the rails increased as the hardness decreased.

The opinion that chemically soft rails would wear best, caused a

* Since this paper was written I have made an extensive inspection-trip over railways in the South and West, and am convinced that for many years light, stiff sections will be of far more value to them than the same weight of metal put into a form not as stiff, but apparently of greater wearing-capacity.

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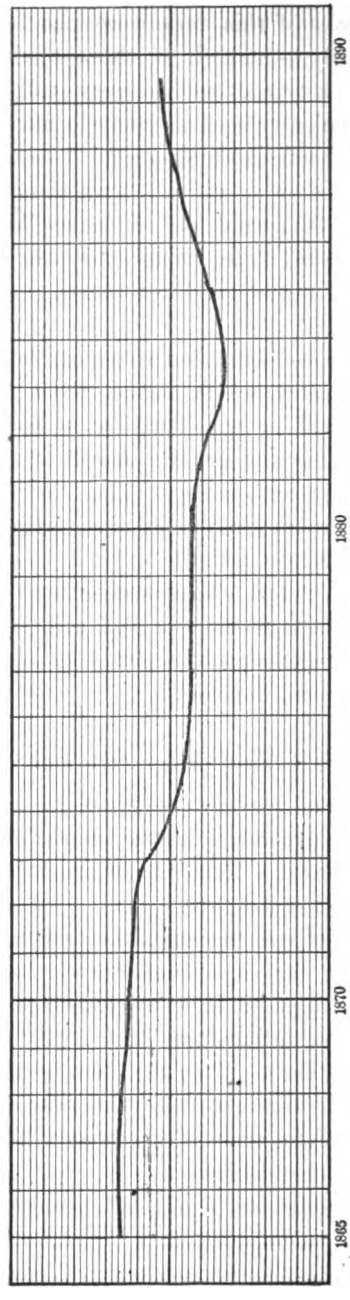

FIG. 6.

marked reduction in the physical hardness of rails from many mills in the output for 1882, 1883 and 1884.

Where railroads are laid with a uniform quality of rail, experi-

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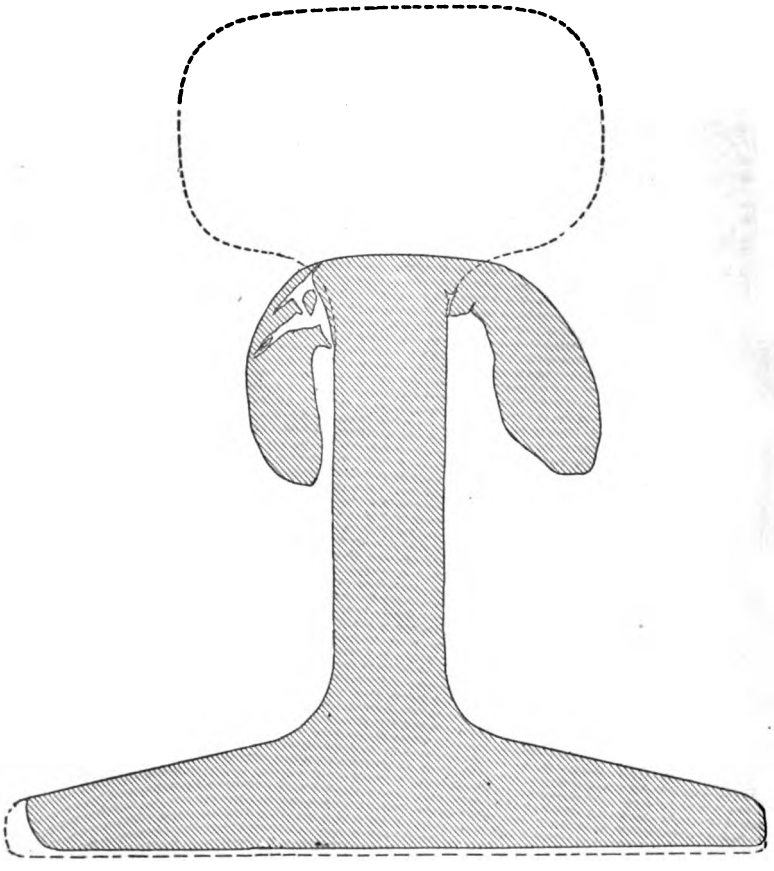
ence has now shown that the wear is not uniform per mile, but is modified by different topographical features, curvature, character of road-bed, classes of motive power, stopping and starting of trains at stations, etc. At connecting-points, large yards and terminals, the drilling for the distribution of cars and making them up for out-

FIG. 7.



going trains, doubles the movements of locomotives, and rapidly increases the wear at such points, as shown by Fig. 7.

The section of rail shown in Fig. 7 was taken from one used at the entrance of the freight-yard of the N. Y. C. and H. R. R. R. at East Buffalo. It was in service eighteen months, being permitted to remain in the track until the sides of the head were bent to the web of the rail. Usually rails require replacing at the same point

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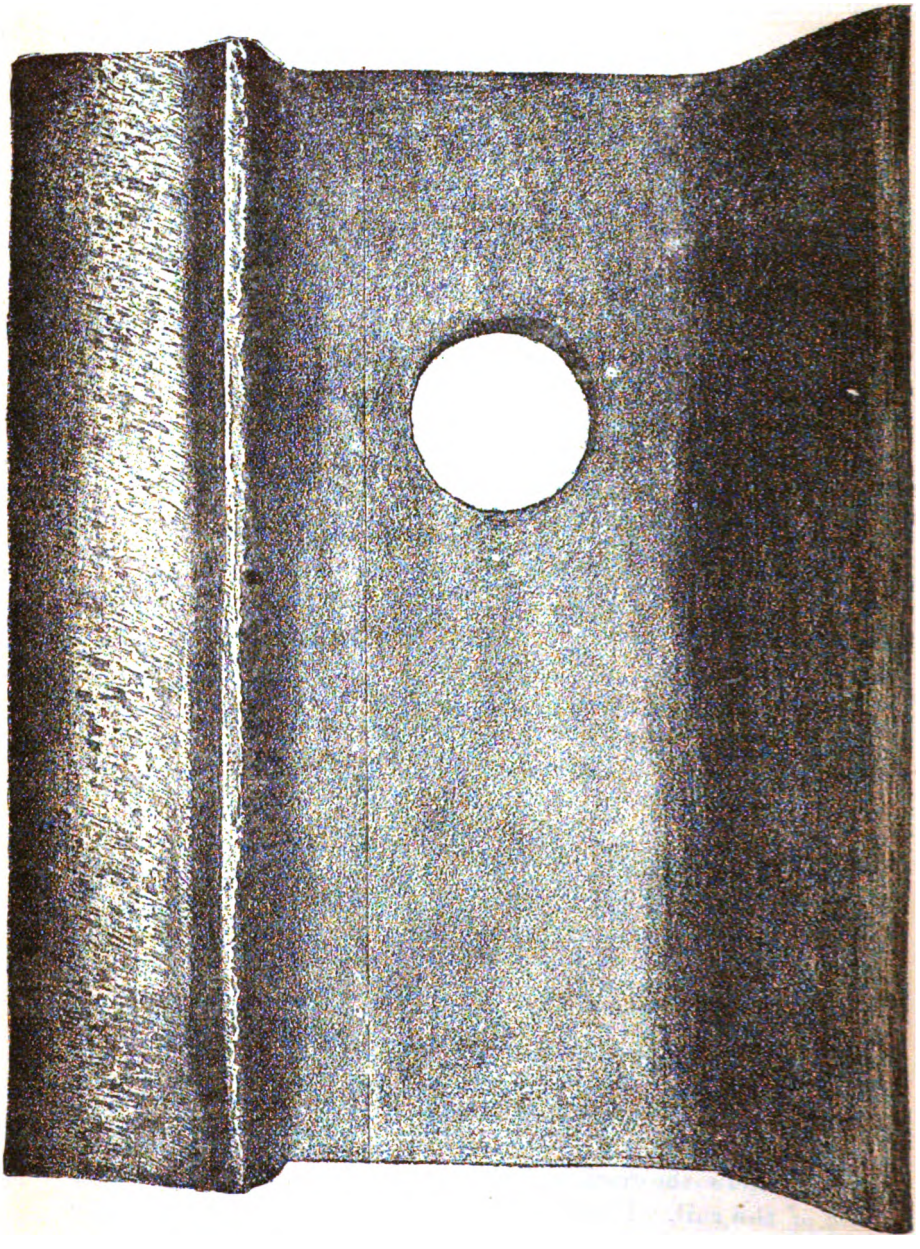
yearly. Rails from same lot, after thirteen years' service in the main track, are still in fair condition. Two and three years is the life of many rails at the entrance of large yards.

FIG. 8.



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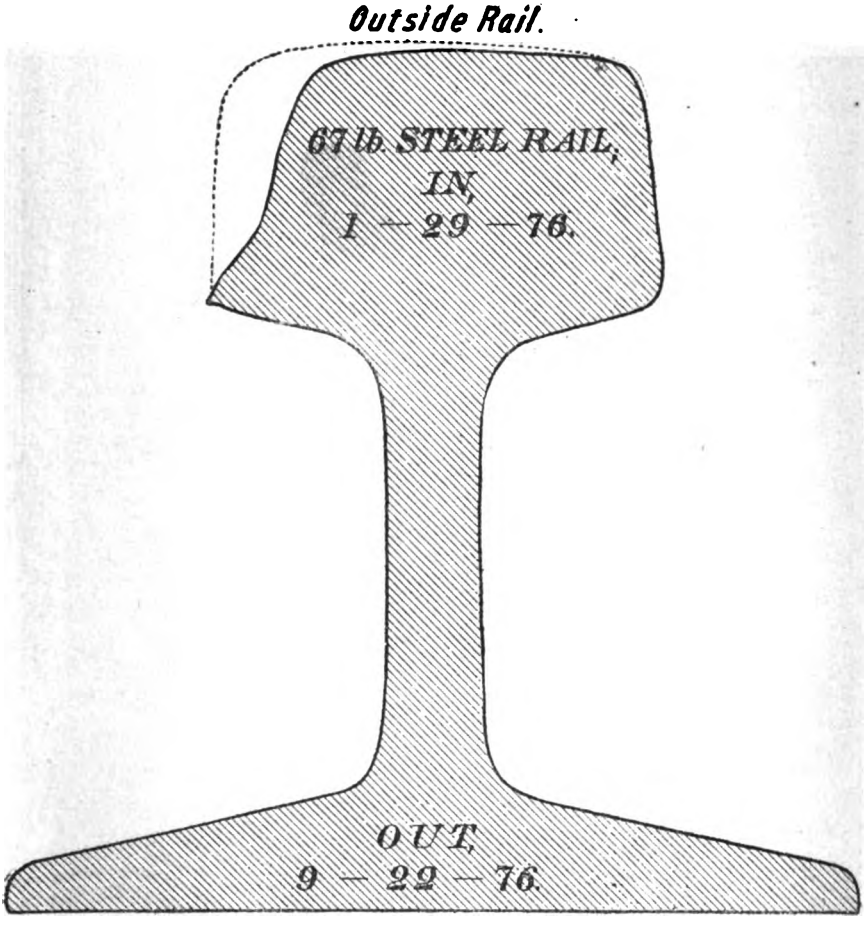
According to the diagrams of several roads in regard to the renewal of rails, the least wear takes place, as a general rule, upon the tangents away from stations, increasing on the gradient tangents. On curves, the rate of wear is even greater than on the gradient tangents, and greatest where curve-gradients occur.

FIG. 9.



Transverse Section of Fig. 8.

The wear on rails on curves is of two classes: 1st. The ordinary wear on the surface of the rail, augmented by longitudinal and transverse slipping of the wheels. 2d. A severe abrasion or cutting on the inner head of the outside rail. (See Figs. 8, 9, 10 and 11.)

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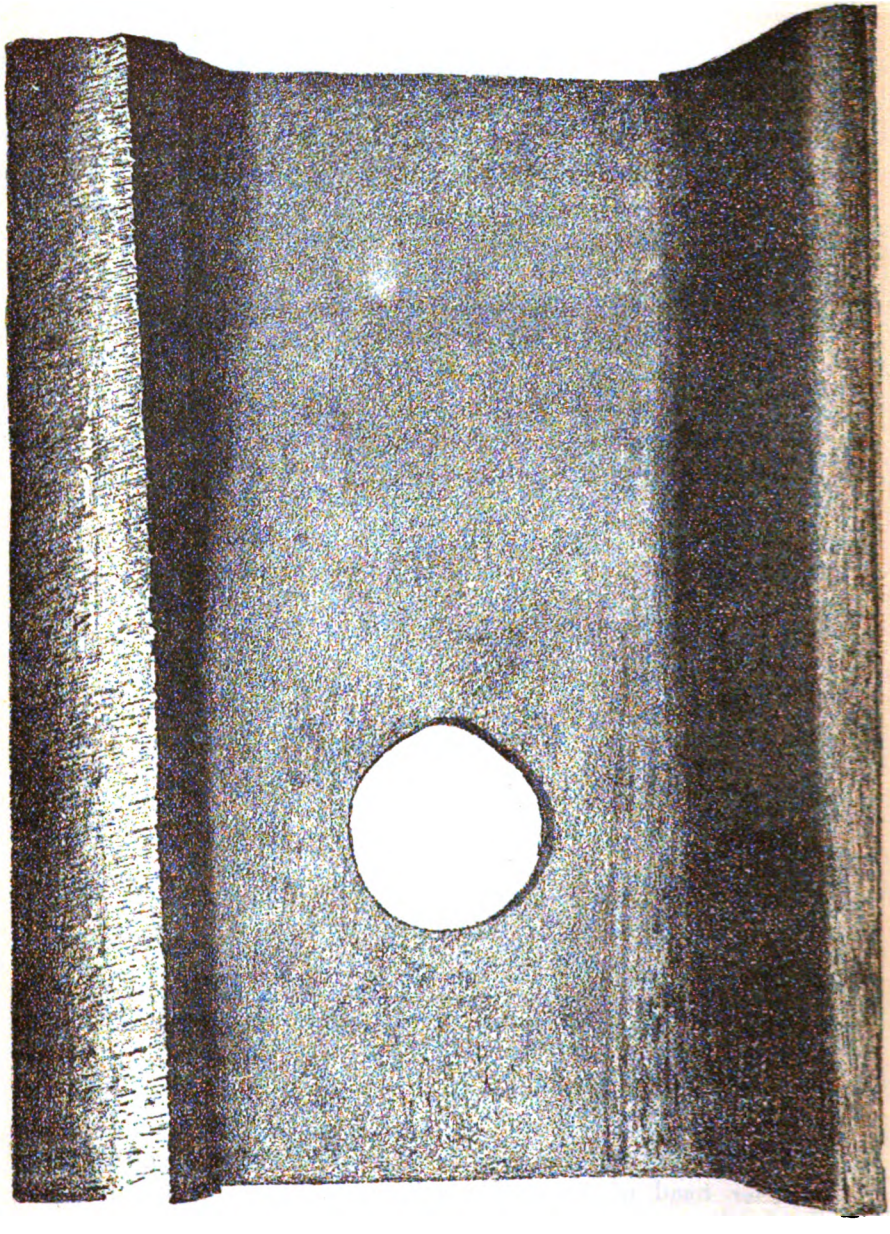
The wear on different roads is not the same, because different systems of track-laying and different joint-fastenings are employed.

I have designed the rail-sections, to which attention is here called,

FIG. 10.



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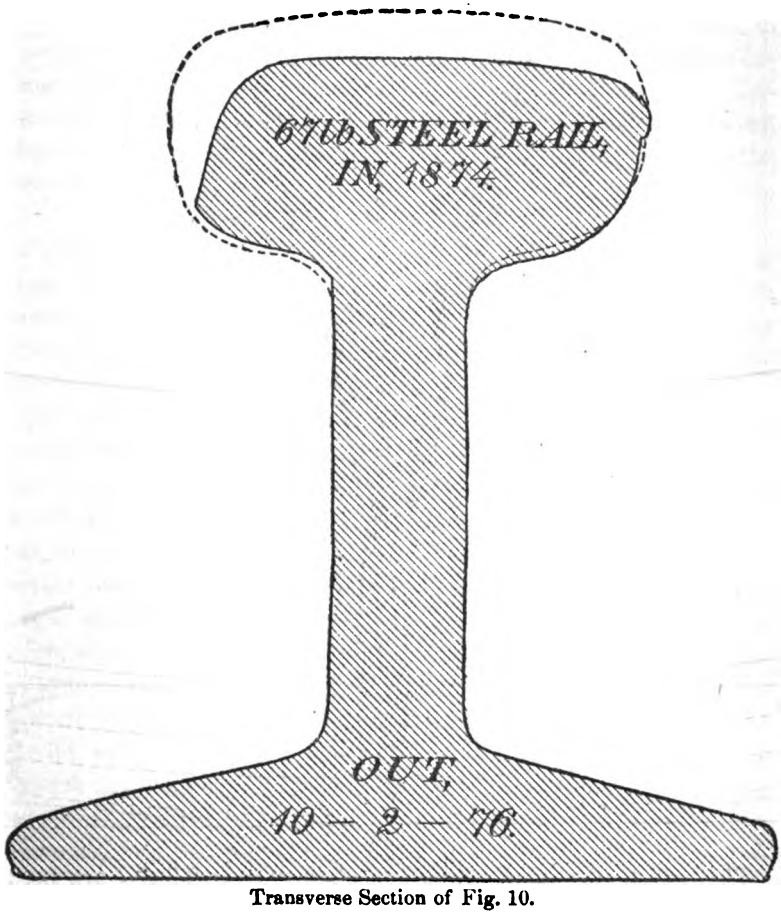
in series of three each, to provide for this unequal wear on the various portions of the road. The rails of each series are designated by letter, and the weight of each section, increasing by 5 pounds each, is further designated by a prefix to its letter indicating its rank in the series, viz.: the first section of the series is called "light," the

FIG. 11.



Transverse Section of Fig. 10.

second one "medium," and the third "heavy." The respective designations will be rolled on their webs in manufacture.

The light sections of series A and B are made especially stiff for the weight of metal used, having also broad heads, well supported by large curves, giving a far greater wearing capacity than might at first seem likely when comparing them with rails having deeper but

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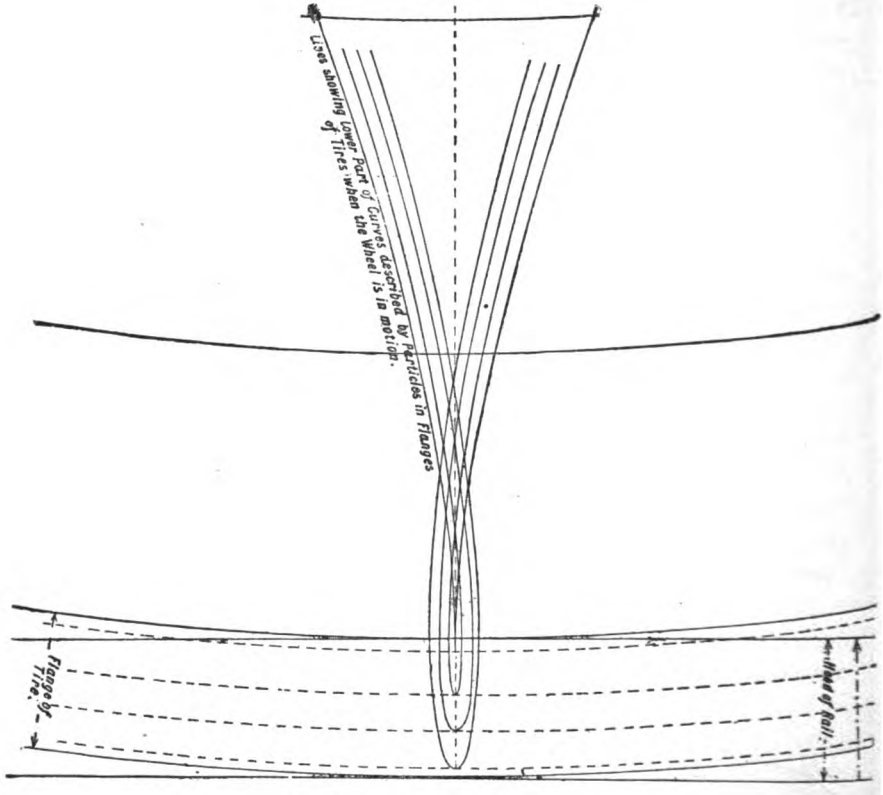
not so well-supported heads. The medium section of the series has a slight increase of width of head, but far greater wearing capacity, *i.e.*, a greater amount of metal to lose per yard to provide for the increased severity of their location under the same tonnage. The stiffness of the section is also increased to sustain the service better, reduce the wear upon the ties, and equalize it more nearly with the rate upon the levels.

FIG. 12



This will be far more important where "treated" ties are used, and also with metal ties, which will come into use on our large lines within a short time.

The still stiffer- and broader-headed "heavy" section for the curves provides a very effective method for reducing the now excessive abrasion of the inner head of the outside rail of the common type on curves. See Figs. 8 and 9 of a rail of the common type, showing severe abrasion; Figs. 10 and 11, ordinary abrasion.

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This statement of the fact of reduced abrasion is not only based on a theoretical consideration of the case, but a large experience in observing the results of such a system in service.

Fig. 12 shows the curtate cycloids described by four selected points in the flange of a wheel rolling on the rails. When the axle is at right angles to the rails on tangents, or radial to the rails on curves, the abrasion is slight. If, however, the axle is inclined to the rail, so that the flange of the wheel touches the rail forward of a perpendicular from the center of the axle, then the abrasion increases with the degree of inclination. (See Figs. 8 and 10.) Increasing the bearing-surface of the rails on curves makes the worn wheels especially run more nearly radial to the curve, lessening the abrasion.

Proposed Sections.

In designing the sections of rails, I have made five series, giving fifteen sections of twelve different weights, ranging from 50 to 105 pounds, for ordinary steam railroads, and four series of light rails for mining, agricultural, manufacturing and construction purposes. The latter are not included in this paper.

By looking at the drawings of the sections, it will be noticed that the inclined surfaces of the section are given by ratio of length of base to the height of the perpendicular, instead of in degrees. The line of the radii for the sides of the webs is above the center of the web, so as to make the lower portion of the web thicker for four important reasons and purposes, viz., 1st, to more nearly equalize the heat of the section between the base and the head, permitting colder rolling and lessening the amount of cold straightening required; 2d, to lower the neutral axis, better equalizing the strain of the metal between the base and head, and checking the tendency to permanent set; 3d, to increase the moment of inertia of the section; and 4th, to check the tendency of the web to bend near the base of the rail under heavy traffic.

The bases of the rails are all thick and of less width than the height of the section, designed so largely for the purpose of permitting the sections to be made as physically hard and tough as the first rails used. Rails made for a long service of years must have a heavy base to provide for wear and oxidation.

Series C, D and E will be used largely upon metallic ties or upon tie plates, where they will be held firmly upon the bearings.

Observations on the broad-headed 80-pound rails of the N. Y. C. & H. R. R. R. show that they have far less tendency to roll in the

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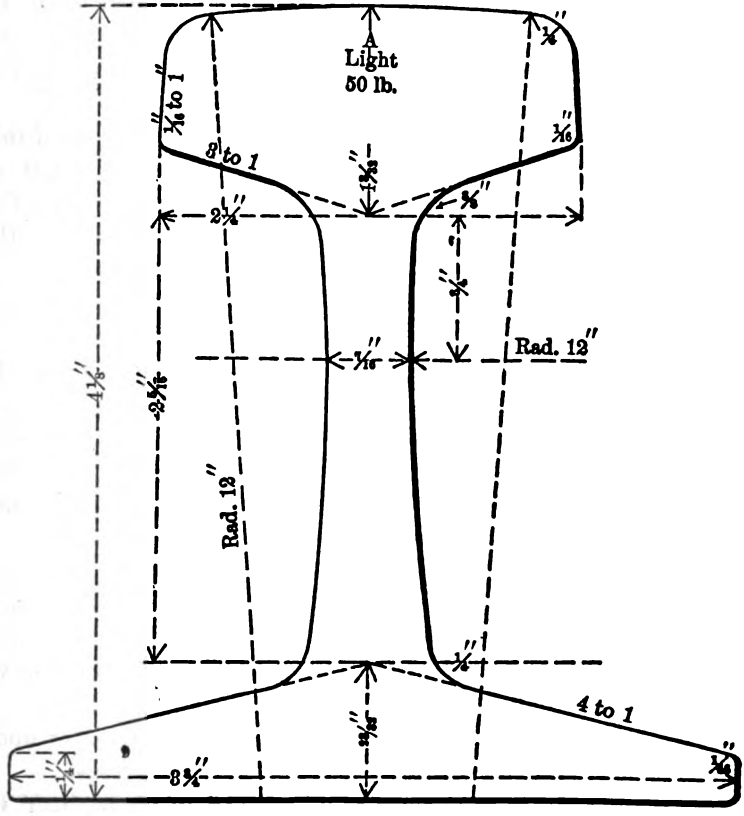
track than the 4½-inch rails with narrow heads, having bases 4½ inches wide with thin edges. On tracks laid with the latter, it is common to find the spikes, particularly those on curves, cut from ⅛ to ¼ off at the throat. This does not occur with the ⅝-inch edge of the 80-pound rail base, and the latter rail is kept in gauge on the curves with far less labor than the narrow-headed ones.

Widening the base of the 80-pound rail to 5 inches would not improve this feature.

In an inspection with my car on April 11th, before any work had been done in the spring upon the tracks, I found in 150 miles of the 80-pound rails only a few rails in 4 miles that were $\frac{1}{10}$ of an inch wide guage. This is unprecedented for even a fall inspection on narrow-headed rails.

SERIES A.



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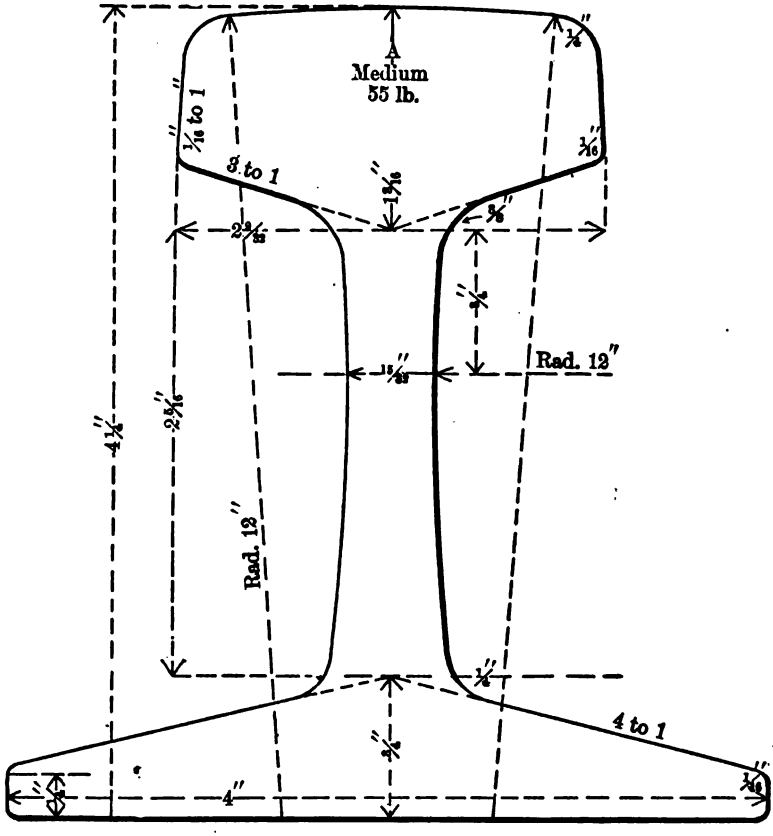

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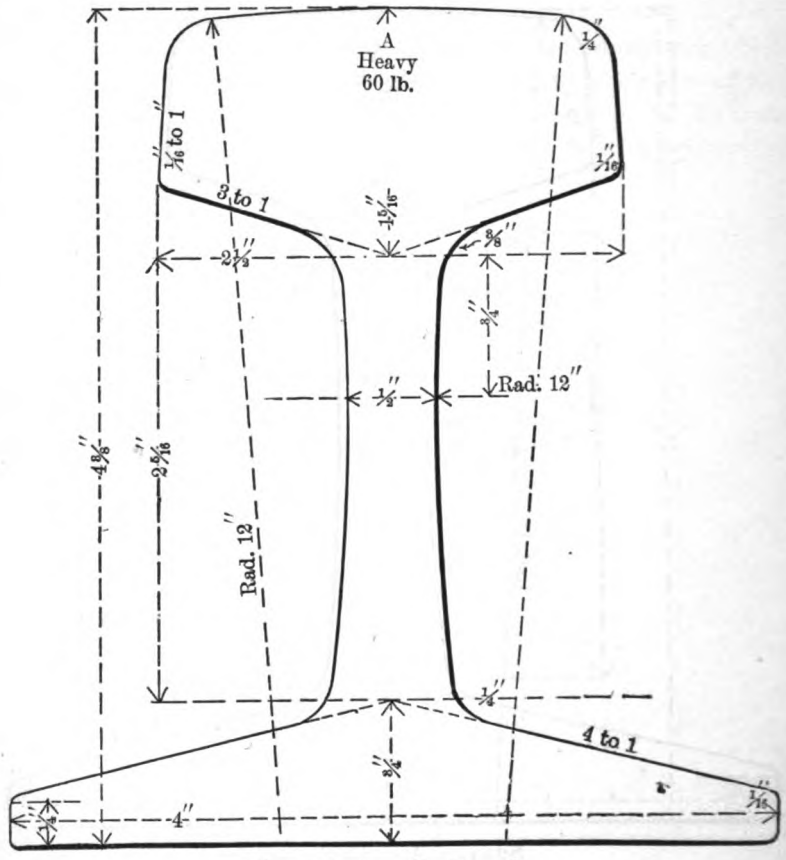

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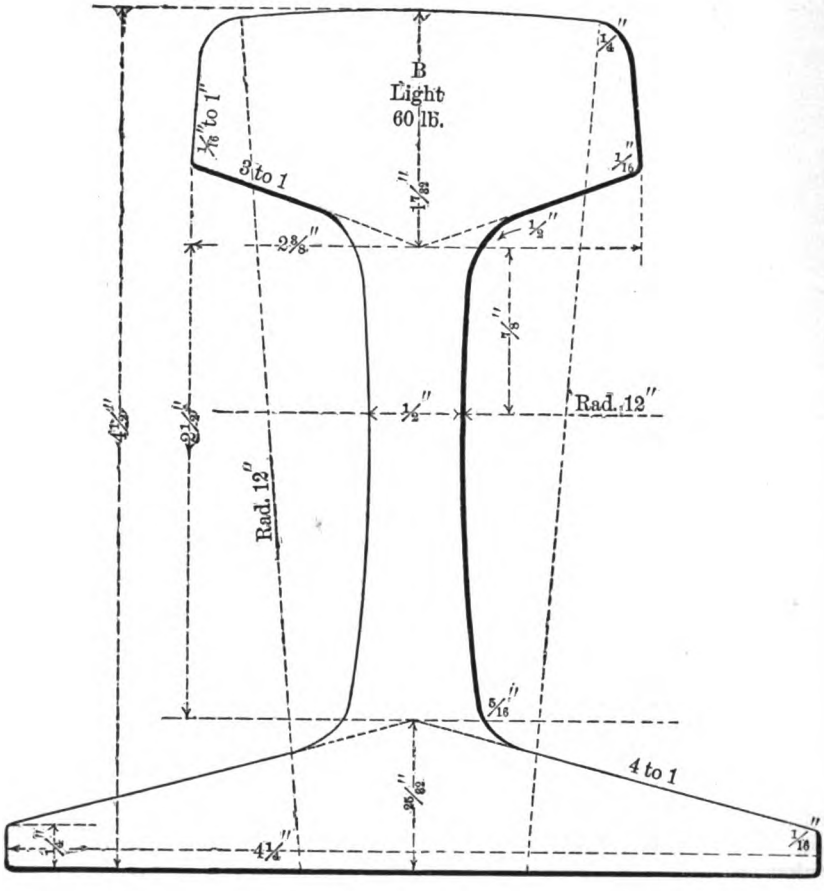

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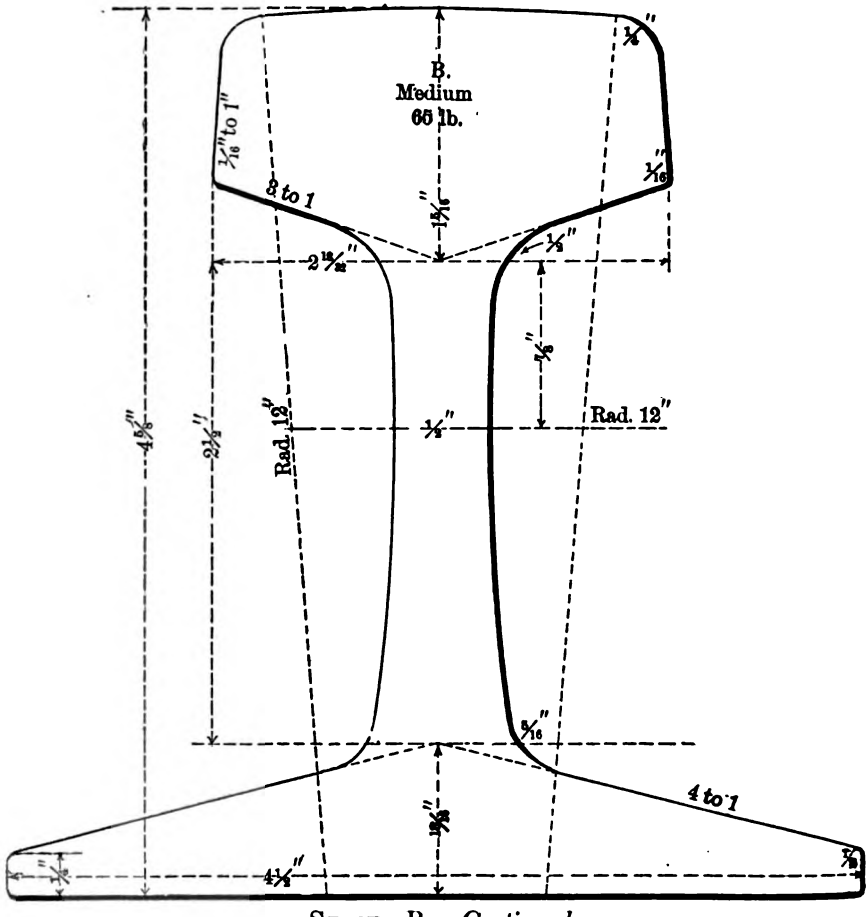

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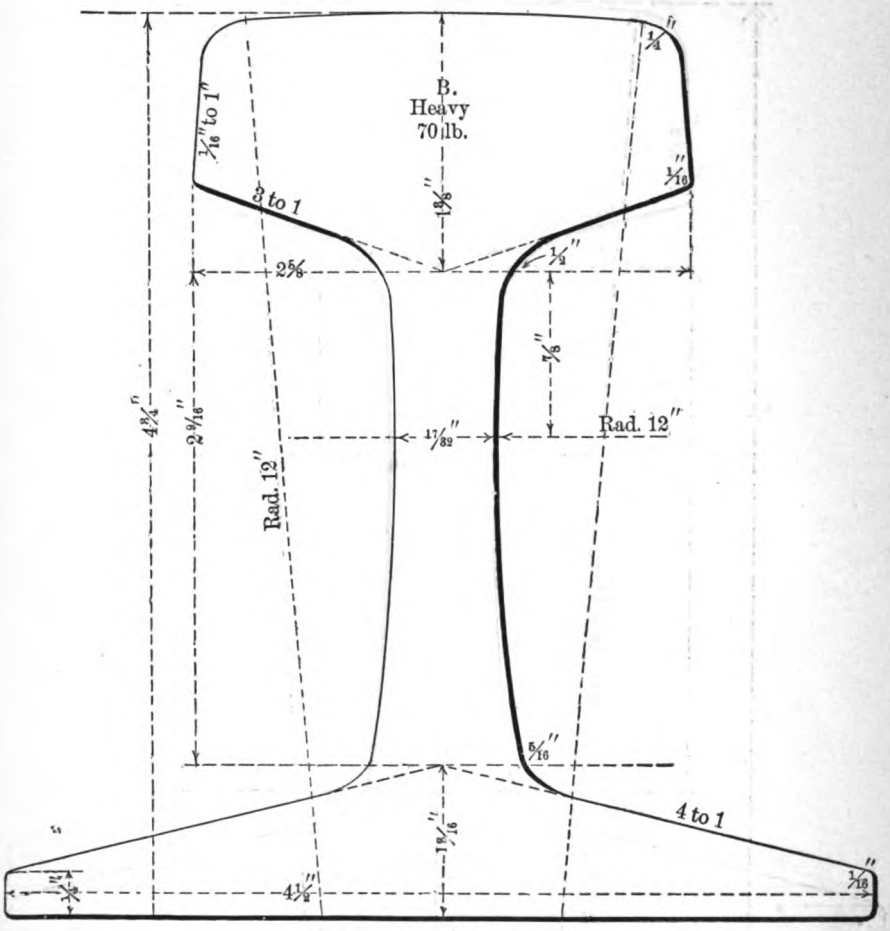

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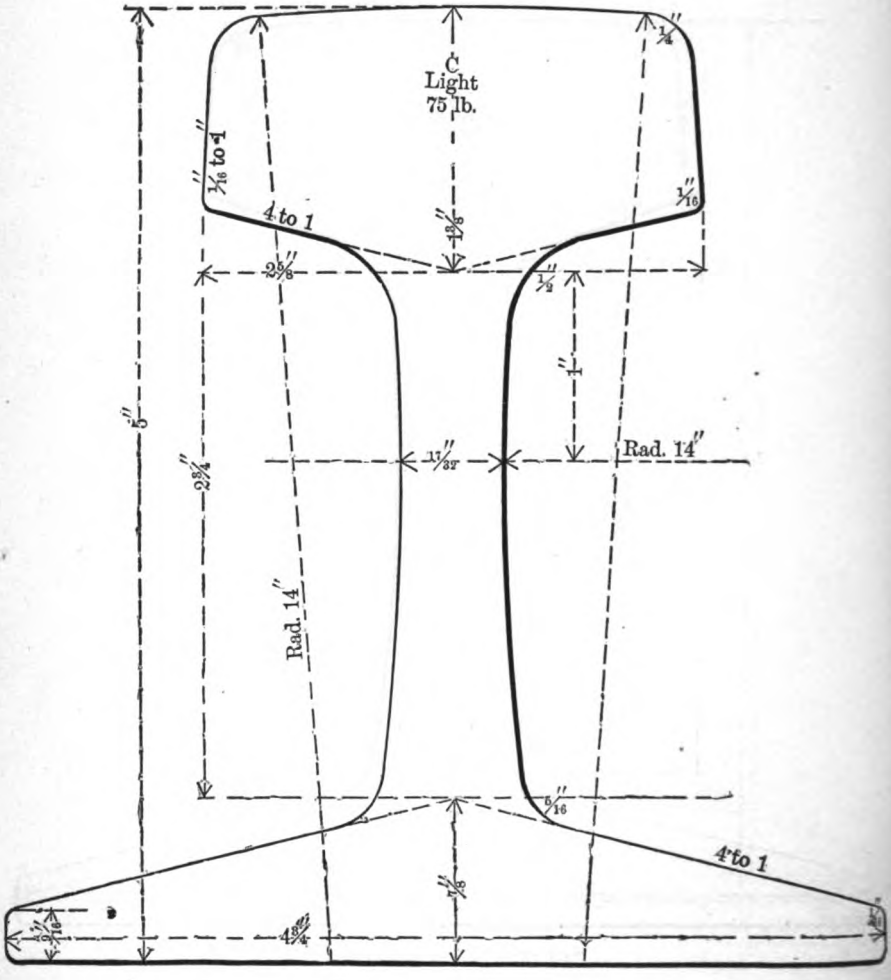

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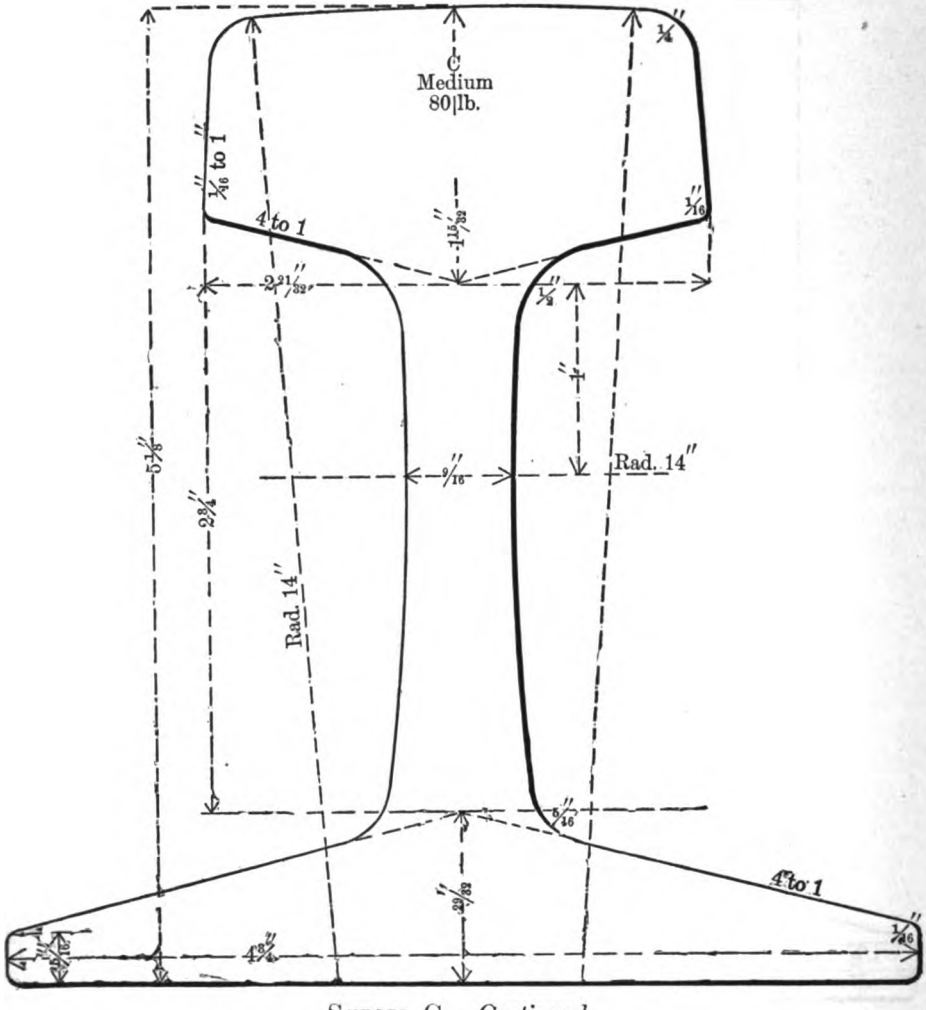

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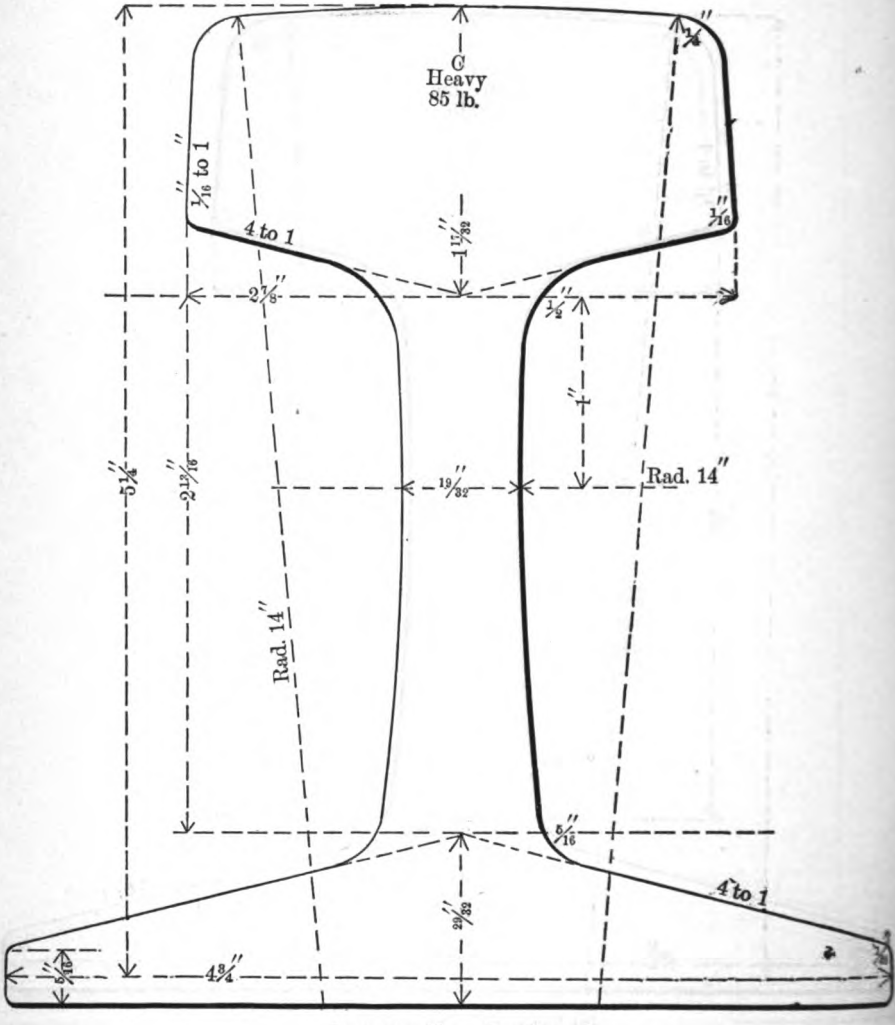

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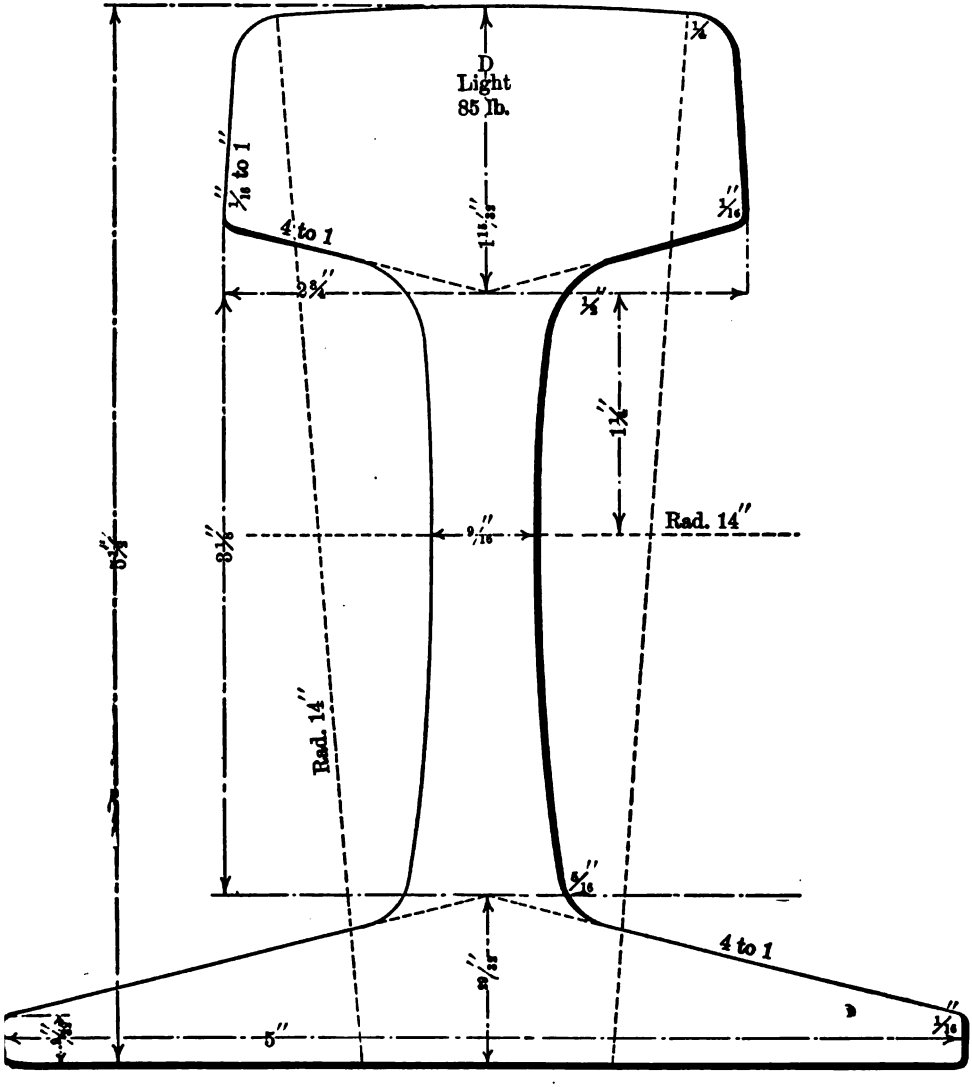
Series D.



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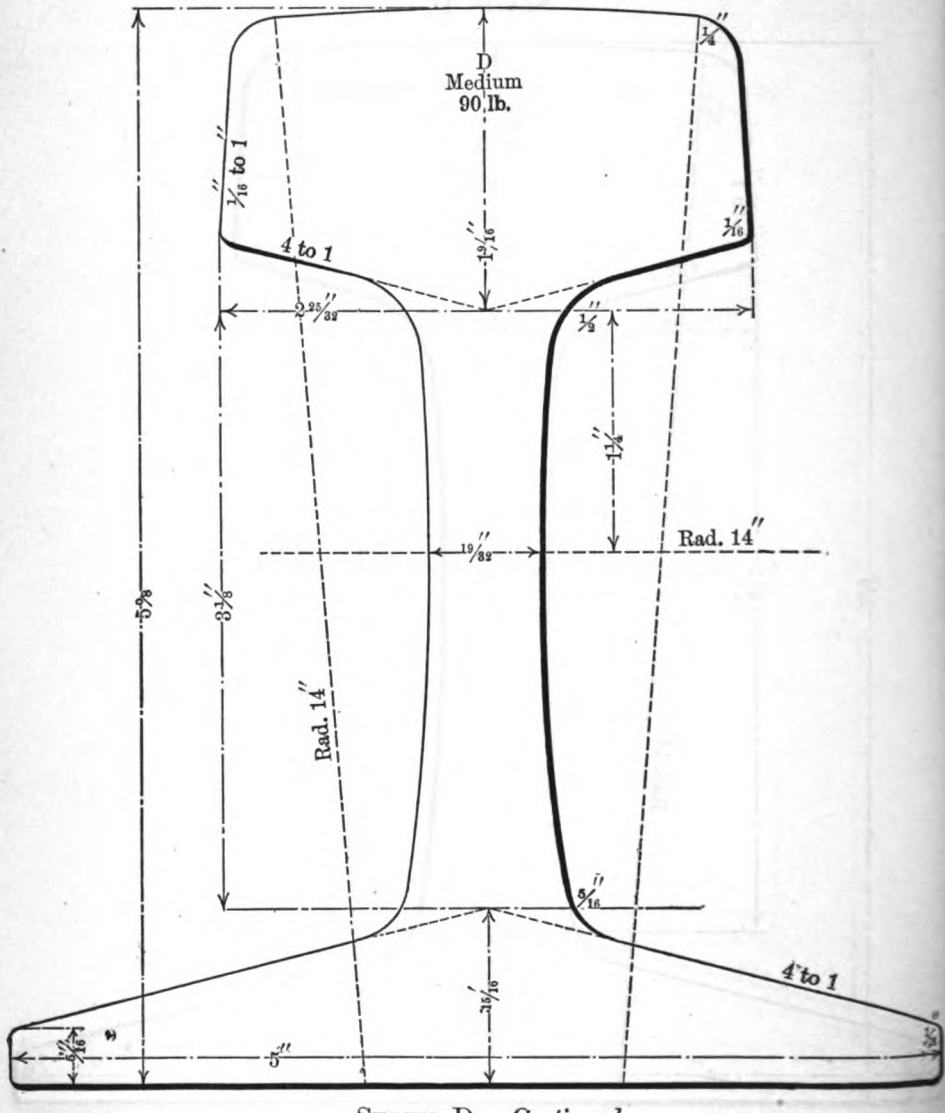

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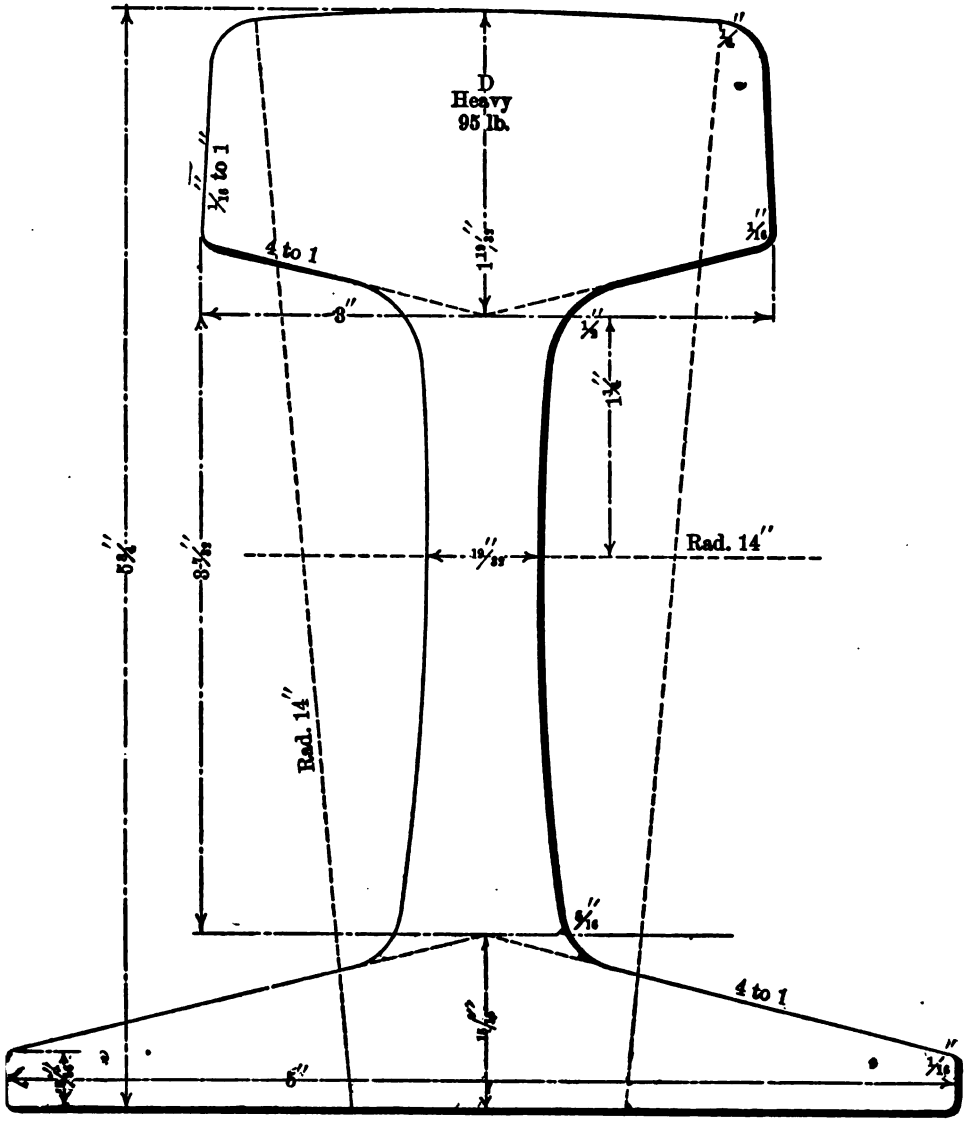

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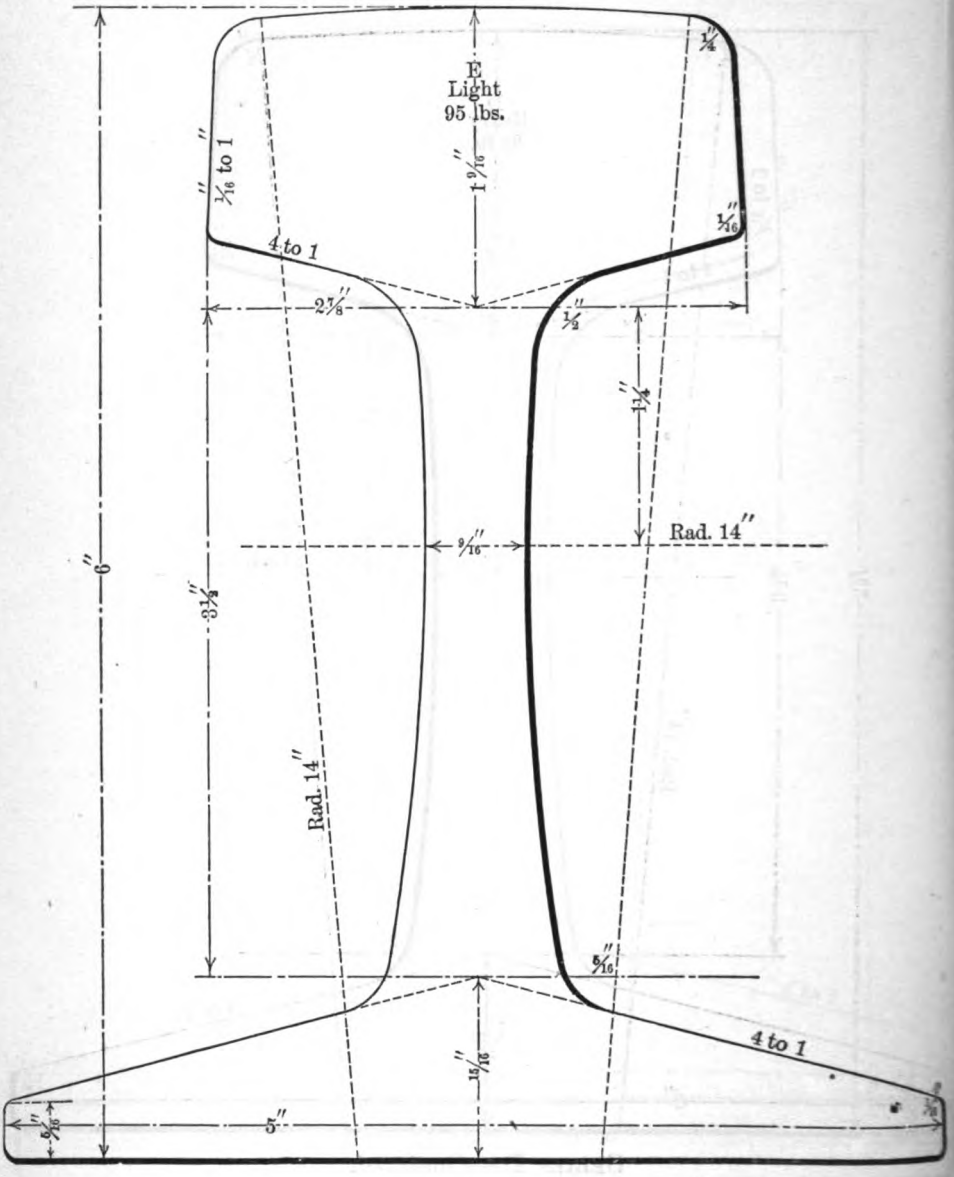
Series E.



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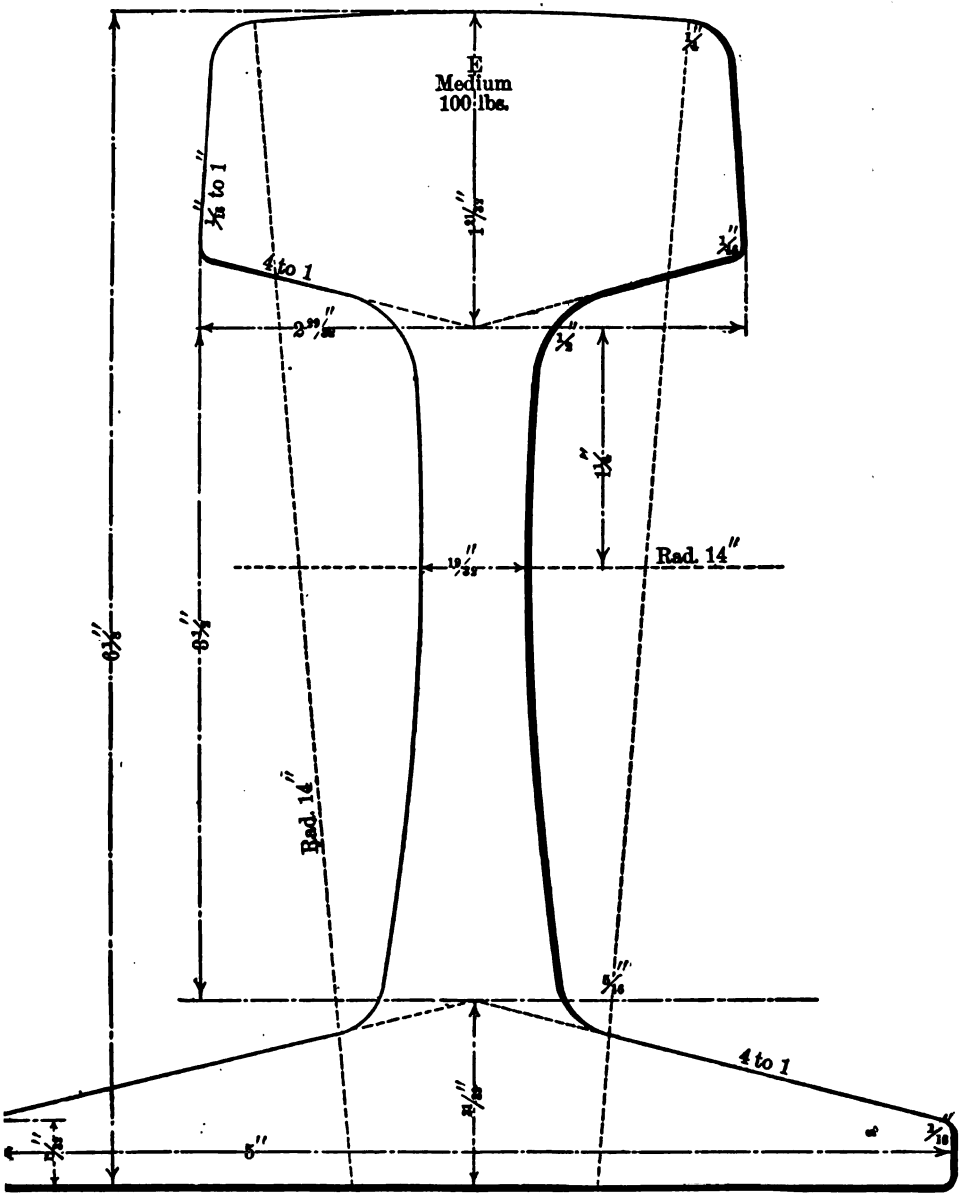

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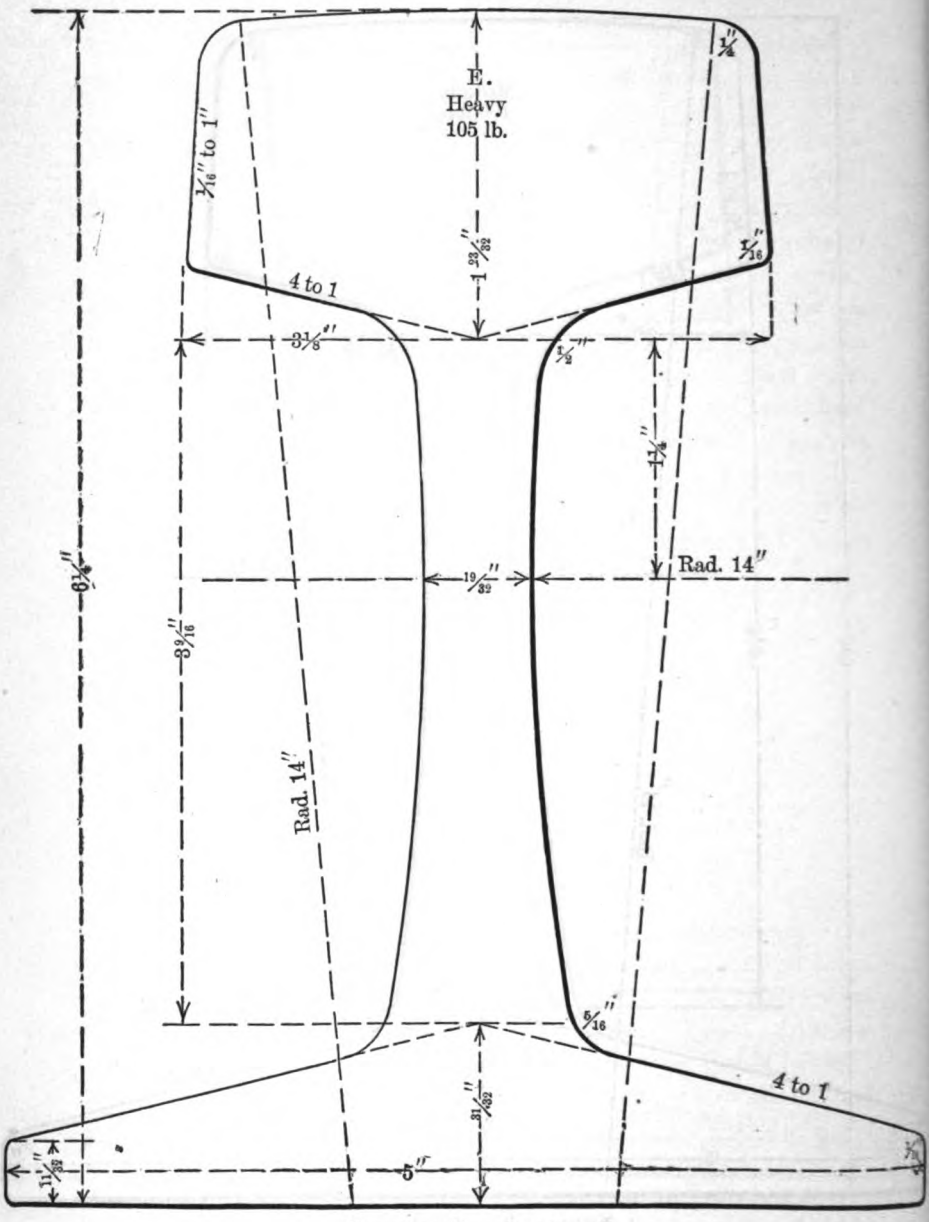

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General Adaptability of the Series or Systems of Sections to all Grades of Railroad Service.

Roads having a light traffic can select the stiff sections of light weight per yard, while roads having a heavy traffic must have the stiff sections, but with more metal for wear. Some roads would only need two sections of a series, while others could economically employ more.

Every section has its advantages and disadvantages, and should be adapted to the general service it can best sustain.

In my system of sections there is a great range of adaptability, not only because it allows a choice of sections, but also because each section permits a large range of service. The heads are broad, of large wearing capacity owing to the efficient support under the heads, are very stiff for weight of metal used, and can be made physically hard, combined with toughness instead of brittleness.

Experiments already made indicate that in physical properties the elastic limits of the steel and the per cent. of elongation in 10 inches could, for the series, be as high as follows:

	Elastic Limits—in pounds.	Elongation. Per cent.
Series A.....	52,500 to 55,000	16 to 20
" B.....	55,000 to 57,500	16 to 18
" C.....	57,500 to 60,000	14 to 16
" D.....	60,000 to 62,500	12 to 14
" E.....	62,500 to 65,000	12 to 14

Tests of rails as nearly similar to those of series A, B and C as could be found, and which have worn well, show that it would be quite safe to adopt the above scale for rails when used in localities where the temperature falls as low as zero in winter.

More extensive tests, covering a greater range of sections and manufacture, may indicate that a higher standard of physical properties can be adopted, or that some change in the contour of the sections may be required.

It is the duty of the engineer to give the manufacturer a standard of physical properties desired, inviting the co-operation of the latter's knowledge in producing a metal of the desired quality.

Attention is called to the deflections shown in tests of two of the

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80-pound rails of different metal. The tests of the first were given in my previous paper.

For light loads, the first rail is the stiffest, one reason being that it is full weight, but the elastic limit of the steel in the head being only 45,680 pounds, it quickly takes a set and wears rapidly.

The tests in this paper are of a rail with actual weight of 78 pounds per yard, but the elastic limit is 62,400, and it does not readily take a set, as will be noticed from the tests, and as far as can be observed is wearing slowly in the track. The difference is not due to difference in sections, but difference in quality of metal.

The deflection-tests, in the postscript to my first paper, of the Delano section, in comparison with the McClure section, are important, showing some difference due to the sections.

Since this paper was presented, I have received a patent (No. 432,159, dated July 15, 1890) for an improvement in railway-tracks, which covers, not the mere sections here shown, but a method of so combining them in a track, according to grades, curves and other conditions of wear, as to secure a substantially uniform duration of service for the whole.

ALUMINUM AND OTHER METALS COMPARED.

BY W. J. KEEP, DETROIT, MICH.

(Washington Meeting, February, 1890.)

SOME explanation of methods of testing will be given before presenting the record of a series of physical tests of aluminum, other metals and alloys.

I.

THE METHODS EMPLOYED AND MEANING OF DIAGRAMS.

The test-bars used, 12 inches long and ½ inch square, are cast in green sand, with chills bedded in such a way that the ends of the bar run against them. By splitting the ends of the bar the depth of chill can be measured. The chills being held in the mould 12½ inches apart, the space between the end of the bar and the face of the chill, after both are alike cold, is the amount of shrinkage, and is recorded in thousandths of an inch per foot.

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Autographic Recording Apparatus.—As bars of some materials shrink more than $\frac{1}{8}$ inch per foot, the bearings for the ends of the bars under test are made of springs 12 inches apart, with sockets at their upper ends. Fig. 1, *a.a.* The bar tested is thus 12 inches long, whatever the actual length may be.

In transverse tests the stress is in the direction of the arrow *b*, causing the bar to bend. The stress is received on the steel receiver *b*, which with *f* is clasped to the bar, and prevents injury to the material of the bar at the point where the stress is applied. The center *d* of the bar moves exactly in accordance with the bend of the bar.

In order to make a record of the position of this center during the whole test, we clamp the cage *b f* to the center of the bar. Behind the bar is an arm connected to the cage at *d*, and the rear end *g* of this arm is fastened to a spring suspended from *c*, while the forward end of the arm carries a pencil *e*, which moves five times as far as the center of the bar. A parallel motion located between *e* and *d* (not shown in the drawing) causes the pencil *e* to travel in a vertical line. All bearings are springs to prevent wear or lost motion. Any motion of *d*, whether from a gradually applied stress, or from a blow, on *b* is exactly reproduced by the pencil *e*. A sheet of paper is held on a flat holder behind the pencil, and can be moved in the direction of the length of the bar, the pencil *e* bearing upon its surface.

The Weight-Testing Machine.—We shall not, in this paper, describe our machines in detail. Those who wish a full illustrated description can procure from the United States Patent Office copies of patents Nos. 399,754 and 403,676. This is a single lever machine, in which by moving a weight out on a beam a gradually increasing stress is applied to the center *d* of the test-bar. The beam is not kept floating, as in ordinary testing-machines, but the center of the test-bar and the beam descend together.

The weight *W* is propelled by a steel cord, and another cord moves the paper which receives the autographic record one-quarter as fast as the weight. The weight and paper can be run either forward or backward. With a 50-pound weight the machine exerts a stress on *d* of from 0 to 600 pounds; with a 100-pound weight on the beam double this stress can be applied. Fig. 1 shows a diagram made by this machine from a test-bar of cast-iron. The test-bar was clamped in the bearings *a a*, and the weight run back until it was over the fulcrum *j*, when no stress was exerted on the test-bar,

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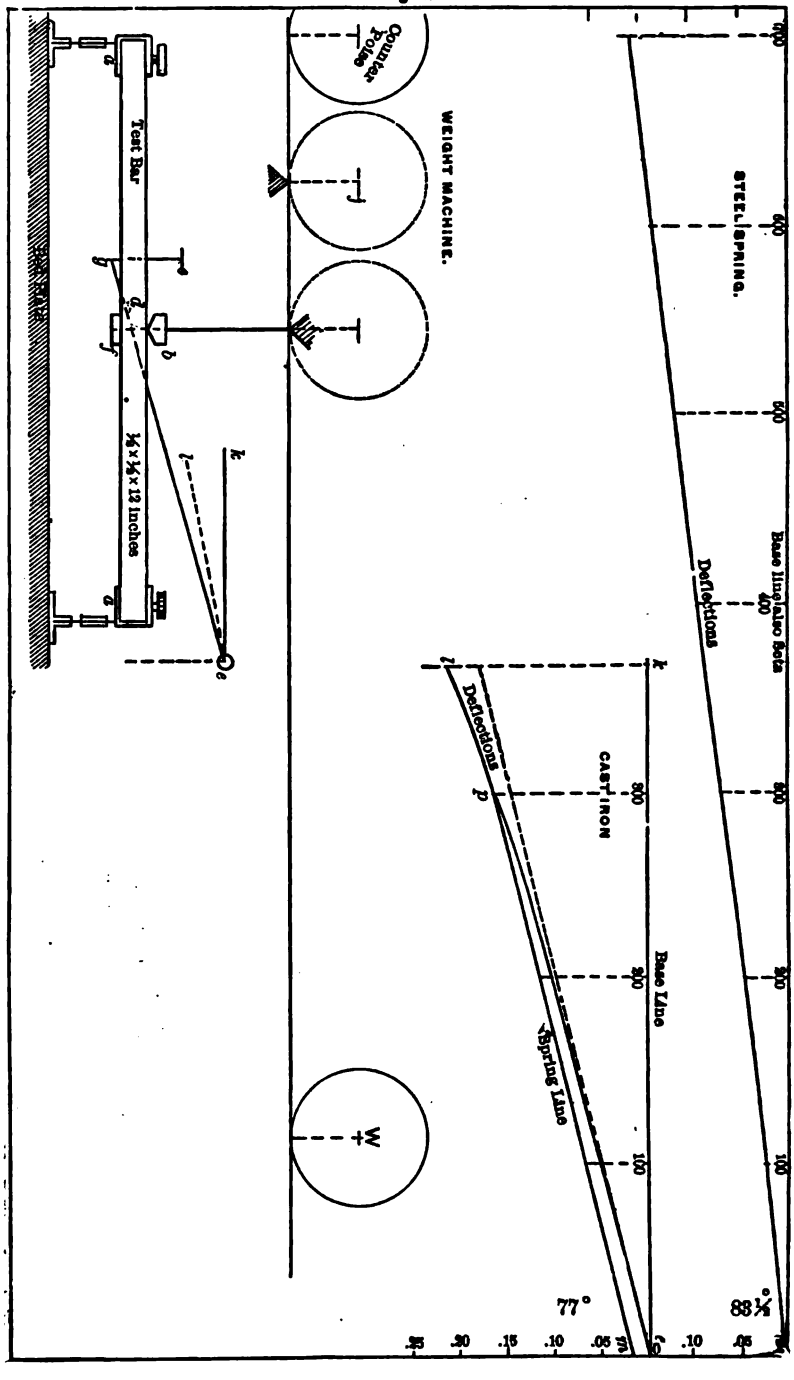
Fig. I.



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the counterpoise exactly balancing the machine, and the pencil, *e*, was consequently at the highest point. The paper was now moved along until the base-line *k e* was drawn. The cord then being attached to the paper-holder, the location of the pencil *e* was marked as zero (0).

As the weight is propelled forward, the increasing stress bends down the center *d* of the bar, carrying with it the pencil *e*. The forward motion of the paper and simultaneous downward movement of the pencil describe a line *e l*, diagonal to the base-line *k e*. By marking off, on the base-line, points corresponding to pounds of stress exerted, and on a perpendicular from the zero-point points corresponding to inches of deflection, we can tell at a glance how many pounds, and how much bending or strain, is represented by any point in the diagonal line or diagram. This is, therefore, a stress-strain diagram. If the test-bar is a perfect spring, the deflections are proportional to the weights, and whenever the stress is removed the pencil will come back to the base-line. Most materials bend, not in proportion to the stress exerted, but faster; and in such a case (illustrated by the diagram from cast-iron) the diagram is a curve. In this case when the stress reached 300 pounds the weight was run back to zero, and the pencil did not come back to the zero-line, but rested at *m*, because this iron, in its original condition, was not a perfect spring. Thus *e m* represents the amount of permanent set that the test-bar has taken in inches under 300 pounds stress. (To measure set or deflection on a diagram made by our machines, we use a scale with 20 divisions to the inch and call the reading hundredths of an inch.) We now run the weight again to 300 pounds and a new diagram *m p* is recorded. If we run back to zero any number of times from the 300 point, the pencil will always traverse the line *m p*, showing that the cast-iron bar has become a perfect spring for all stresses less than 300 pounds. The ultimate strength cannot be influenced by repeated stresses, if each is less than this proof-stress.

The character of the metal has been changed, but if the weight is run beyond 300 pounds the instant it passes this point the original character of the metal again appears and the diagram is a continuation of the original one, and, taken as a whole, is exactly the same as it would have been if the stress had been continuously increased and not relieved at 300 pounds. In Fig. 1 the curve *e p l* should have been a continuous curve. The engraver has by mistake made a depression at *p*. The same mistake is made at *e*, in Fig. 2. All spring-lines from the same bar during one test (and perhaps through all tests) are parallel.

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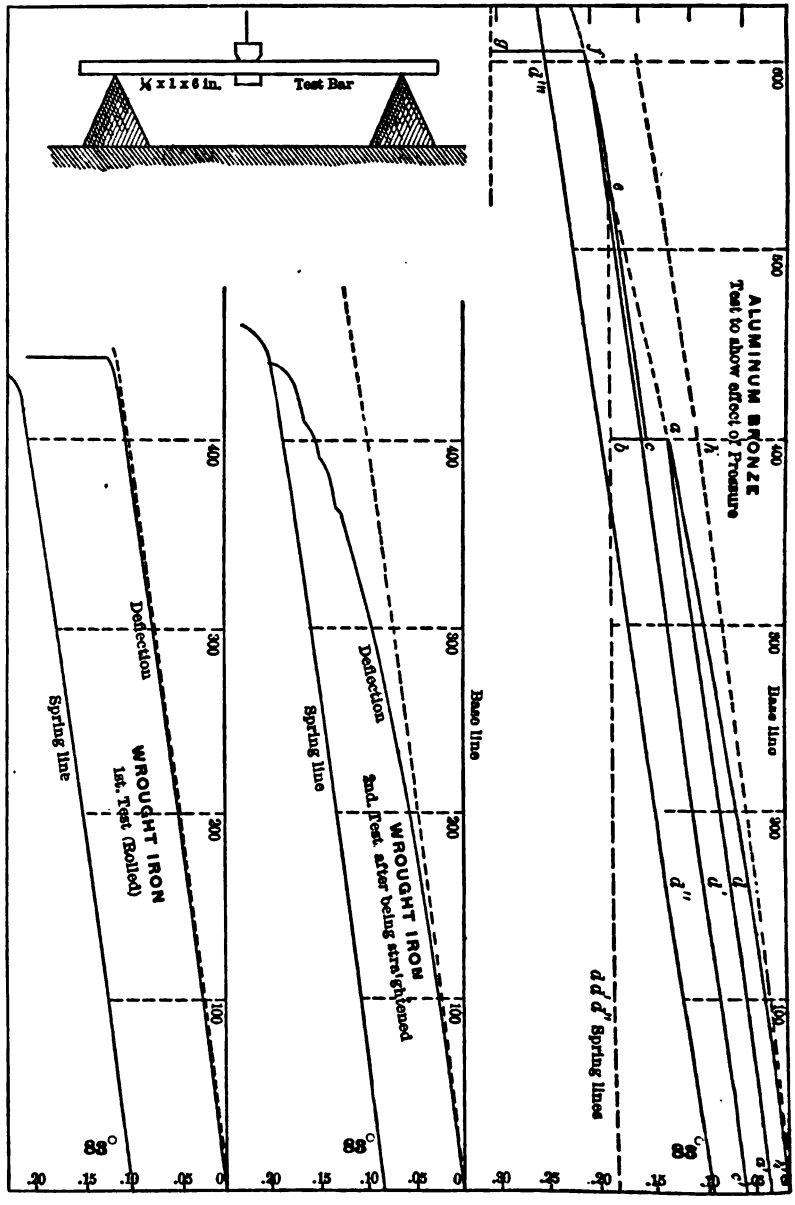
To show more clearly this change in the nature of a metal by work done upon it (which illustrates the effect of forging, rolling and wire-

Fig. 2.



drawing), we produce in Fig. 2 a diagram from aluminum-bronze. Here we ran the weight to 400 pounds and then made a spring-line

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$a\ a'$. When we again reached 400 we knew that if we should continue moving the weight out on the beam the line of the original diagram $h'\ a$ would be continued to e, but we stopped at 400 and bore down on the beam with one hand, deflecting the test-bar to b. On allowing the bar to recover it came back only to c. We then ran the weight from c out beyond 400 and found the diagram to be a spring-line until it reached e, which we found to lie on a line running parallel to the base-line and passing through b.

After passing this point e the original diagram was resumed and continued until f. We then made a new spring-line $c'f$. On again reaching f we again bore down to g and the bar returned to d'''. We then ran the machine back to the zero-line and made the spring-line $d''\ d'''$, and on pushing the weight beyond d''' we found the spring-line continued until it intersected the ordinate through g, when the original diagram was again resumed.

We conclude from our observations that for gradually applied stress or a dead load a bar is a perfect spring for all weights less than the greatest proof-load that has been applied, unless the proof-load was very near the stress which would cause the bar to give way, in which case the deflection of the second diagram, just before reaching the proof-stress, may exceed that of the first diagram and the lines may not meet.

The set which each of the bars shown in Fig. 2 has taken, differs from the set taken by the cast-iron of Fig. 1. The set in the cast-iron is caused by a change in the shape of the molecules of the metal. The set in the bars of Fig. 2 is caused by the flow or sliding on each other of the molecules of the metal which become partially viscous when a certain stress has been exceeded. Until flow sets in the speed with which the weight is run out on the beam of the machine has little or no influence upon the form of the diagram. When, however, the bar begins to bend rapidly it is difficult to refrain from sympathy with the specimen and to slow up, which allows the metal to flow still more and thus produces a steeper curve. This is seen in Fig. 2 in the test of wrought-iron after straightening.

Most metals which take a set as the stress is increased, but which do not flow, probably take set, however small may be the stress applied, though the amount of set may be too small to be measured. Metals which can take no set, but which flow when a given stress is exceeded, may form a perfect spring and therefore take no set up to the commencement of flow.

It will be seen that the spring-lines of cast-iron and of many metals are curves, and yet the fact that the pencil always traverses the same line proves it to be a spring-line, though the peculiar

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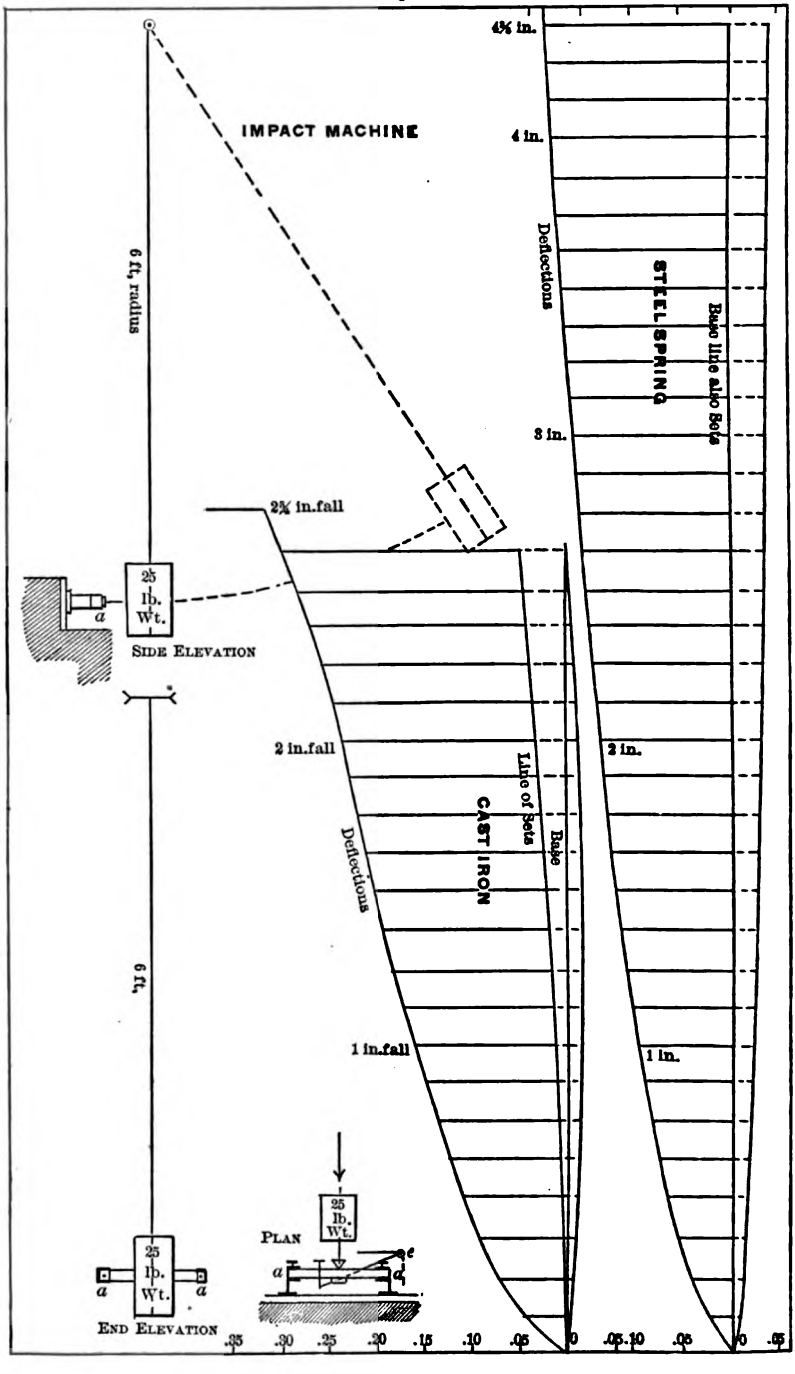
Fig. 3.



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internal spring of the atoms of the metal does not cause the deflections to be exactly proportional to the stress.

Good illustrations of this are the diagrams taken from a bar of rolled wrought-iron, Fig. 2. The bar was a perfect spring for all weights less than 400 pounds. After that it took a slight set and at about 430 pounds gave way completely, the metal exhibiting viscosity to a remarkable degree. Reversing the machine as quickly as possible we made a spring-line. It will be noticed that the bar in this case gave way at an earlier point than under the first application of stress.

We now straighten the bar and test it again. The bar does not act at all as it did during the first test. It takes set at each increment of stress and after 300 pounds it flows rapidly, but the metal does not give way suddenly at 430 as before. We take a spring-line and notice that it passes the former strength. This often occurs and is due not to any rest given the bar but to the work done on the bar.

The Impact-Testing Machine.—In this machine, Fig. 3, the test-bar is fixed in the same bearings of the same autographic recorder, as was already described; the same pencil and paper are used, and all conditions are alike in both machines, except that, in the impact machine, the stress at the center of the bar is a blow from a 25-pound weight swinging on a radius of six feet. The paper is moved ahead about $\frac{3}{16}$ in. after each stroke to remove the record of the stroke just made. The first blow is struck with $\frac{1}{8}$-in. fall, and each succeeding stroke has $\frac{1}{8}$ in. more fall than the preceding. The pencil records deflection and set and also the vibration of the bar.

We show in Fig. 3 an impact diagram from the same cast of cast-iron as the bar of Fig. 1. Gray cast-iron takes set at each blow, and often the set increases more rapidly as the point of fracture is approached.

Much of what we have said regarding weight-diagrams applies to impact-diagrams also. If the test-bar is a perfect spring, the set-line will coincide with the base-line. If a material is tested with a certain number of blows and flow has not begun, and a certain amount of set has been taken and the test is repeated from the beginning, no set will be taken until the last blow of the previous test is reached, for the bar is a perfect spring for all blows less than the proof-blow. This is especially true of bars that have been rolled down to the $\frac{1}{2}$-in. square, provided they have not been bent and straightened previously.

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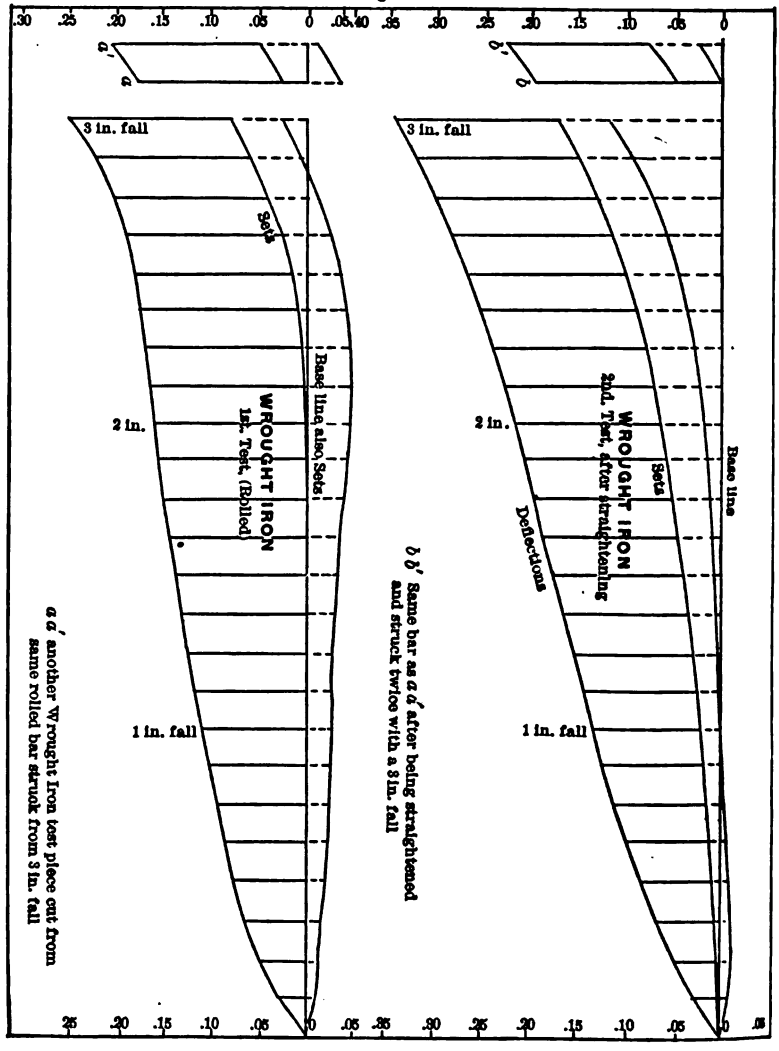
The impact-diagrams shown in Fig. 4 are from a test-bar cut from the same rolled bar as the bar tested by weight, the record of which we had in Fig. 2. It takes no set until after a 2-inch fall. If we had stopped at $1\frac{7}{8}$ in. and had repeated the test, all diagrams

Fig. 4.



up to this point would have been alike, for the bar, up to the 2-inch fall, is a perfect spring. When we reach a blow that causes the metal to flow, each blow simply forces the bar down more and more, adding to the set and total bend; beyond this point we find the effect of the blows to show only in the sets.

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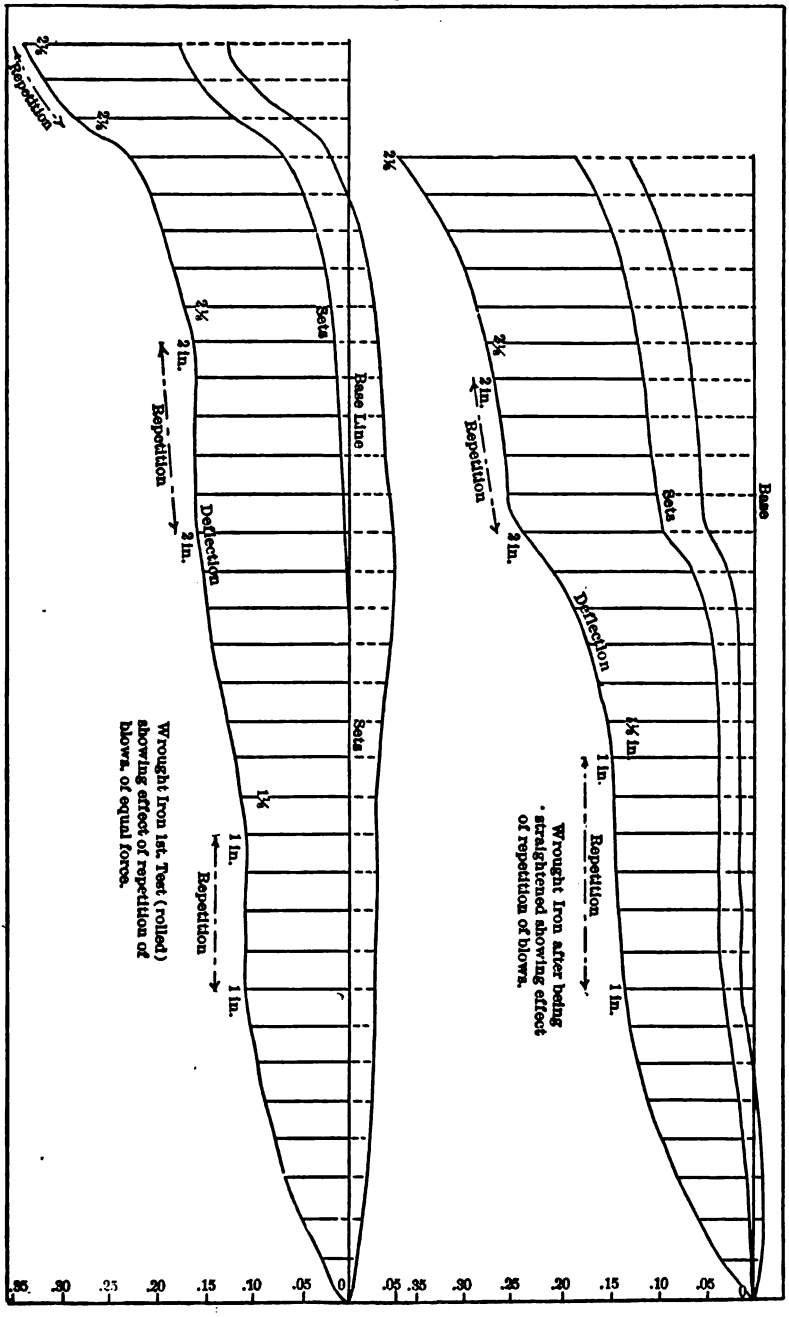
Fig. 5.



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We took another bar cut from the same piece and struck it two blows from a 3-in. fall; and we give the record at the top of the former record in Fig. 4. We now straightened each of these bars and repeated each test. The whole character of the bars had changed. Impact shows this change more fully than weight. The bars took set at each blow.

To observe the effect of a repetition of the same blow, we cut from the same piece of wrought-iron another test-bar which we tested, as before, up to a 1-in. fall, and then repeated this blow four times. The lines recording deflection, set and vibration, as shown in Fig. 5, are all parallel, showing no effect on the strength of the bar.

The regular blows were then struck up to a 2-in. fall, when the metal began to flow. Five repetitions of this blow depressed each of the lines of the diagram at each blow. This is much more apparent in the repetition of the $2\frac{7}{8}$-in. blow. The bar was now straightened and tested again, when it was found that each blow, whether a repetition or not, depressed all lines of the diagram.

Before flow has set in in such metals as we have just considered, and in many cast metals even after set has been given to the bar, a repetition of the same blow does not seem to diminish the strength of the bar.

The question of crystallization from repeated blows cannot be raised in any ordinary test. In our tests the blow is prevented from affecting the material at the surface of the bar. When repeated blows are given to a bar by a sledge on the face of the metal, each blow may develop the same force, but because of the effect of the blows upon the crystals of the metal at the point of contact, the metal is finally wedged apart. Such is not a legitimate impact-test. To compare weight and impact, we should compare bars which are of exactly the same constitution. As, for example, diagrams from two test-bars cut from the same straight rolled bar of ingot-iron or wrought-iron, or two exactly similar cast bars poured from the same ladle of metal. Or, take two pieces of wood $\frac{1}{2}$-in. square, cut from adjacent places in a board. Take the breaking or yielding points of any pair of such bars, one tested by weight and the other by impact, and compare sets, deflection, vibration, weight and inch-pounds shown, and we shall see that the effect of impact cannot be deduced from any data obtained from the weight-diagram or *vice versa*. Each metal tested has marked peculiarities of its own.

By striking each specimen a blow from $\frac{1}{8}$-in. up to the blow which

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causes it to give way, all are treated alike and the records are relative and comparable.

The new conditions presented by the results of our tests necessitate some new definitions for use in the explanation and comparison of our diagrams.

Rigidity or stiffness is the ability of a material to withstand stress and retain its original form. It is *perfect* when any stress that can be applied will not cause the material to change form. The diagram of a perfectly rigid bar would exactly follow the base-line. *No* rigidity would cause the form of a material to change without limit on the least application of stress. The diagram of no rigidity is a straight line perpendicular to the base-line at the zero-point. It is the abscissa on which changes of shape or the amount of strain are measured. *The diagram of rigidity* will be a line starting from the zero point and lying diagonally between the ordinate (base-line) and abscissa, which meet in a right angle at the zero point.

The measure of rigidity is the angle that the line or diagram of rigidity makes with the abscissa. It is called the *rigidity angle.*

In all diagrams made by our weight-machines the scale on the ordinate is either 50 or 25 or 16.66 millimeters for each 100 pounds, the scale being selected which will show the qualities of the metal to the best advantage.

The scale of deflections on the abscissa never varies, but is always multiplied by five to make the divisions easily seen and to make the error five times less.

All *rigidity angles* are reduced to the 25 millimeter standard and marked in degrees.

To reduce from the 50 to the 25 mm. scale take the difference between the angle and 90° and subtract this difference from the rigidity angle recorded. The resulting angle will be the rigidity angle for a 25 mm. scale.

For example, the angle of the diagram for cast-iron for a 50 mm. scale is 83° ; for a 25 mm. scale it would be 76°.

For the half-inch square steel spring the angle for a 50 mm. scale is 86¼° ; for the 25 mm. scale it would be 82½°.

Rigidity, Hardness, Brittleness and Temper.—Not only are some materials more rigid than others, but by alloying metals of low rigidity a compound may be produced which shall have a high rigidity. Rigidity is a structural property of matter and is not due to hardness.

With the sample of tool-steel from which our diagrams were made, the rigidity is the same whatever be the temper or the hardness.

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When given a dark-blue spring-temper it withstood various applications of stress both by weight and impact up to 800 pounds and up to a 5-inch fall without any effect upon the strength or shape of the bar. The bar was then hardened as much as possible and tested by weight to 400 pounds when it broke. The rigidity of the hardened steel was exactly the same as when blue. In both states the steel had the same bend and therefore the same elasticity up to 400 pounds, and yet the hard steel was brittle compared with the blue steel. We may properly say, then, that a metal need not be rigid to be brittle, nor does varying brittleness involve necessarily varying rigidity. Rigidity is not an element of strength or weakness, but it is a very important element in a structural material. It cannot be determined by the ultimate strength or by the total deflection, but only by making a spring-line and measuring the angle of rigidity.

Stress is measured on the ordinate or base-line from the zero-point in pounds avoirdupois for the weight-machine, and for the impact-machine by the number of the blows or by the value of each blow in inch-pounds.

Deflection is the distance (at right angles from the base-line) traversed by the center of the bar under any given stress. It is measured in hundredths of an inch from the base-line, the scale being laid off on the abscissa passing through the zero-point.

The stress diagram may be a straight line if the deflections are proportional to the stresses (Hook's law), or a curve if the deflections increase more rapidly than the stresses. The diagram may conform to Hook's law for a certain distance and afterwards become a curve. The point where proportionality ceases is usually called the *limit of elasticity*. It is the limit of perfect elasticity, but a better term for it is *the limit of proportionality*.

Elasticity is the property by virtue of which a body tends to regain its original volume or shape after it has been distorted. Elasticity is perfect or imperfect according as volume or shape is wholly or only partially regained. Elasticity of shape is the only elasticity considered in the testing of materials of construction, and in our tests only so far as it relates to the recovery of the central point in a test-bar one-half inch square and 12 inches long, supported at the ends and subjected to stress applied at the center in a direction at right angles to the length.

The measure of elasticity is the distance that the center of the test-bar moves towards regaining its original position on the base-line when the stress is removed.

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The length of this line is given in hundredths of an inch, and is measured on the zero (or a parallel) abscissa. If the center of the bar returns to the base-line, the bar is perfectly elastic, and the diagram will generally be a straight line. In such a case the measures of elasticity and deflection are the same.

We can call the measure of elasticity *elastic deflection.*

For convenience we shall call the straight line conforming to Hook's law a *spring-line,* as we denote the rigidity by the angle this line makes with the zero abscissa.

When the deflection is not proportional to the stresses, the additional deflection will not be recovered when the stress is removed. Such part of the deflection as is recovered is the *elastic deflection,* and the portion not recovered we may call *set deflection* or *permanent set,* or simply *set.*

To divide the total deflection, as produced by the weight-machine, into elastic and set deflection, we proceed as already described, viz.:

When we think that we have applied nearly as much stress as the bar will stand, we run the machine back to zero. If the bar has received any set the pencil will not come back to the zero of deflection, although it rests on the abscissa marking the zero of stress at a distance from the ordinate equal to the total permanent set.

If we now run the machine forward we shall describe a new stress-diagram, in which the deflections are proportional to the stresses, and it is therefore a spring-line, and its angle is the angle of rigidity. If now we lay off a line parallel to this spring-line and tangent to the original diagram at its zero-point we can see that between the base-line and this plotted spring-line all deflections are *elastic deflections,* and that between this spring-line and the original diagram all deflections are *set deflections.*

From the fact that it took the same stress to produce the diagram that consisted of both elastic and set deflections, and that which consisted of elastic deflections alone, it is evident that it takes no more force to produce both elastic and set deflection during an original test of a bar than it takes to produce only elastic deflection during a subsequent test. The bar has not necessarily been strengthened (although this is generally the case), nor has it become more rigid, for its rigidity angle is always the same, whether it has received 100 or 400 pounds stress, or whether the test was the first or was made after straightening; but up to the proof-stress it will take no set, and has only elastic deflection.

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Resilience.—The definitions of this term, as given by various writers on the strength of materials, cover the idea of the work that can be done by a body, which has not taken set, in recovering its original form after stress has been removed; in other words, the work done in producing elastic deflection.

Some writers say that the area of the diagram bounded by the base-line, the spring-line and the abscissa, measuring elastic deflection at the point of greatest stress (*i.e.*, one-half total stress multiplied by elastic deflection), represents the total resilience. Some even multiply one-half stress by the total deflection in cases where set has been taken, as for example, in tests of cast-iron. All these writers seem to consider that the length of the line measuring elastic deflection depends upon the stress exerted to produce this deflection. For this reason we have endeavored to give a definition of elasticity that shall be entirely independent of the stress that produces the elastic deflection.

We doubt the advisability of using the term resilience in a general way, for, although elastic deflection is proportional to the stress in each given material, yet the elastic deflection for different materials is not the same for equal stresses.*

Still greater difficulty arises when we endeavor to calculate the resilience shown by a diagram from the impact-machine. When

* To illustrate the misunderstanding that will naturally arise with men who are simply practical, and who do not understand the scientific meaning of "work" and of "resilience," we will refer to two examples: A tempered steel bar, Fig. 1, and a maple bar, Fig. 7, both of exactly the same size, and both perfectly elastic for the stress we shall apply to each, namely, 50 pounds. The elastic deflection of the steel bar is 0.03 inch; that of the maple bar is 0.30 inch. The formula, ½ stress × elastic deflection, gives a total resilience for the steel of 0.75 inch pound, and for the maple of 7.5 inch-pounds, while the most that could be performed during recovery for both springs would be to raise the 50 pounds to the starting point, and the bar that bends most is least suited for structural purposes.

If we reduce the above results to the resilience per pound of each material, we shall find for steel 0.84 inch-pound, and for the maple 0.96 inch-pound per pound of each respectively. This seems still more absurd.

It may be claimed that only bars from like materials can be compared; but when we come to metals, especially iron, we cannot do this; for the chief difference consists in the presence of some other metal or metalloid, and such an alloy cannot be compared with other alloys without such anomalous results as those above stated.

The difficulty arises from the assumption that deflection has direct relation to the weight that produced it, whereas it is the character of the material which governs the amount of deflection which any given stress will produce. The material that yields most readily to stress will give the largest result when stress is multiplied by deflection, and if this result is divided by the weight of the body the quotient will be greater or less according to the specific gravity of the material, which has nothing whatever to do with strength.

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Fig. 6

W. J. KEEP'S IMPACT TESTING MACHINE.

Table showing the number of inch pounds developed by each blow. Also giving an arbitrary value to each blow in pounds avoirdupois to allow comparison and tabulating with records from weight machines.

The value for an inch pound was obtained by testing a good sample of Swedish pig iron which broke in the weight machine with 288 pounds and by impact with a 2⅛ inch fall. The table was constructed from this data.

Inches perpendicular fall of 25 lb. weight	Inch pounds developed.	Value of each blow in lbs. avoirdupois.	Inches fall of 25 lb. weight.	Inch pounds developed.	Value of each blow in lbs. avoirdupois
0.12	3.12	16.94	3.12	78.12	423.53
.25	6.25	33.88	3.25	81.25	440.47
.37	9.37	50.82	3.37	84.37	457.41
.50	12.55	67.76	3.50	87.55	474.35
.62	15.63	84.70	3.62	90.63	491.29
.75	18.70	101.65	3.75	93.70	508.24
.87	21.87	118.59	3.87	96.87	525.18
1.00	25.00	135.53	4.00	100.00	542.12
1.12	28.15	151.46	4.12	103.12	559.06
1.25	31.25	169.41	4.25	106.25	576.00
1.37	34.38	186.35	4.37	109.37	592.94
1.50	37.41	203.29	4.50	112.55	609.88
1.62	40.61	220.23	4.62	115.63	626.82
1.75	43.72	237.18	4.75	118.70	647.77
1.87	46.87	254.12	4.87	121.87	660.71
2.00	50.00	271.06	5.00	125.00	677.65
2.12	53.13	288.00	5.12	128.12	694.59
2.25	56.25	304.94	5.25	131.25	711.53
2.37	59.37	321.88	5.37	134.37	728.47
2.50	62.55	338.82	5.50	137.55	745.41
2.62	65.63	355.76	5.62	140.63	762.35
2.75	68.70	372.71	5.75	143.70	779.30
2.87	71.87	389.65	5.87	146.87	796.24
3.00	75.00	406.59	6.00	150.00	813.18

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the products of two sets of numbers are the same, the mechanical results may not be the same. Hence, if we wish to compare results of tests of impact-machines, we must construct the machines exactly alike in main features. We use, for instance, a 25-pound hammer and swing it on a 6-feet radius, simply because we have selected these proportions—at first arbitrarily—but now because so much work has been done with this machine. A 25-pound weight multiplied by a 5-inch fall gives 125 inch-pounds. A 5-pound weight falling 25 inches gives a product, also, of 125 inch-pounds, as in the first case, but the effect upon the test-bar is not at all the same.

Comparative Elasticity of a Bar Under Stress by a Dead Weight and by a Live Load.—By examining diagrams from both the weight and impact-machines, we find that a test-bar is generally more elastic under a blow than under a stress gradually applied.

It is not correct to say that equal deflections represent equal impact-stresses; for we have already shown that elastic deflection is not dependent on the amount of stress, but on the character of the material.

If we examine a large number of diagrams from such a material as cast-iron, we shall find that some will stand a greater and some a less number of blows, as compared with the stress which caused fracture by the weight-machine; but in all cases it will be found that the elasticity is greater under impact than under weight.

In impact-diagrams the diagram of sets lies next to the base-line and the diagram of total deflection outside this line of sets.

The measure of elasticity is the distance between these two diagrams. The measure of sets is the distance between the base-line and the diagram of sets.

On the right-hand side of the diagram of sets is a diagram showing the vibration of the bar, and the measure of vibration is the distance between the diagram of vibration and that of sets.

In the impact-diagram, when the bar is a perfect spring, there will be no set, and the diagram of sets will, therefore, coincide with the base-line.

For the present, until further investigation shall suggest a better means of comparison, we give in Fig. 6 a table prepared by giving an arbitrary value to an inch-pound.

Proportions of the Test-Bar.—Testing bars of the same material, but of different dimensions, brings to light the fact that having tested one bar of a material the data thus obtained cannot be used (with present formulæ) to correctly calculate the strength of bars of

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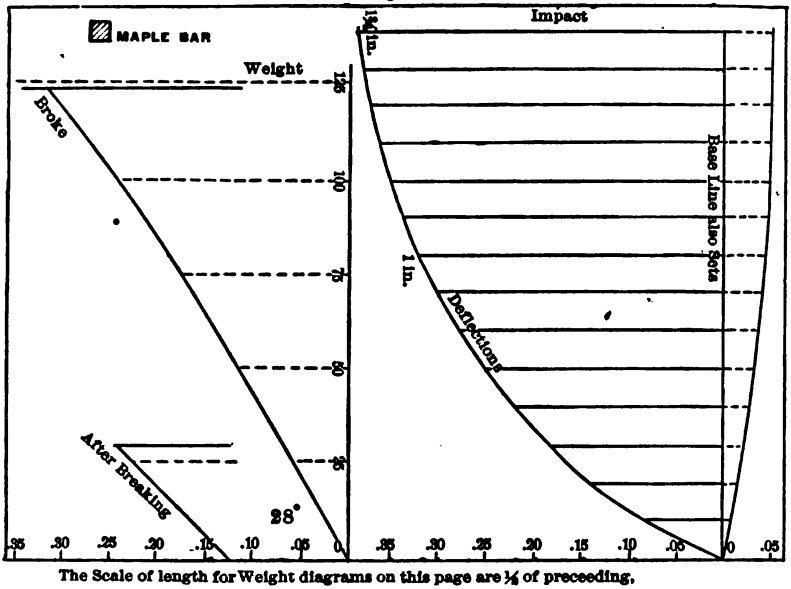
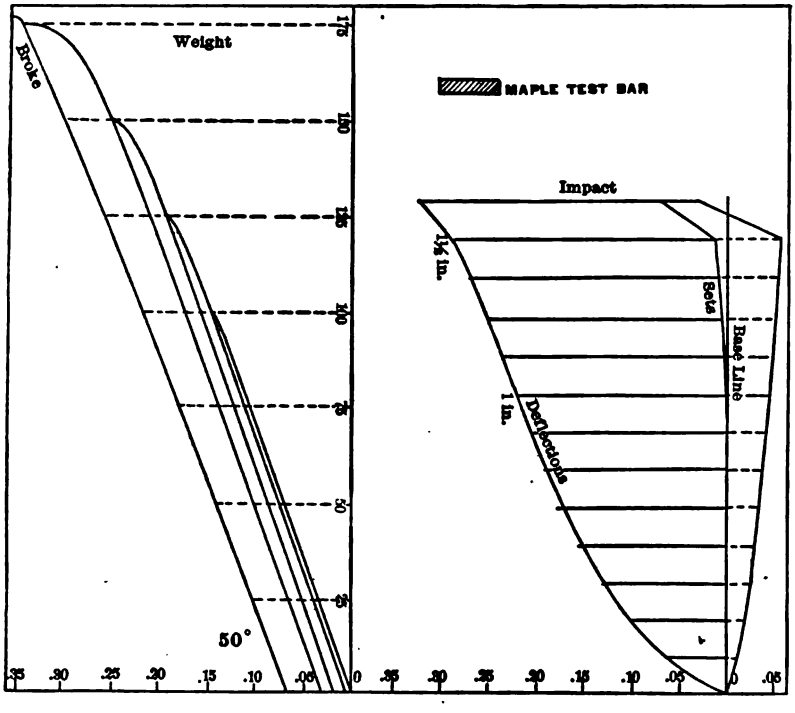
Fig. 7.



The Scale of length for Weight diagrams on this page are ¼ of preceeding, but Scale of Impacts are the same



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other dimensions. It is generally understood that the transverse strength is directly as the breadth, and as the square of the depth, and inversely as the length.

If this be true, a bar 6 inches long, ¼ inch deep and 1 inch wide (see Fig. 2) should have the same strength, and should make the same diagram as a bar ½ inch square and 1 foot long. The diagrams from maple-bars of the dimensions just mentioned, given in Fig. 7, show that the above proposition does not hold. In the tests reported in other parts of this paper, we have in several cases given diagrams from bars of both the above proportions, which show the same thing.

It is not, therefore, possible to reduce area to any given standard; as, for example, to *one* square inch cross-section of test-bar, if a bar of another section has been tested, or to a standard length if any other than such standard length has been used in testing. We, therefore, take a bar ½ inch square and 1 foot long and give all results as they are recorded without any reduction. This is, in the main, necessary; for there is no way by which we can express deflection, elasticity, set, rigidity or hardness, etc., as so much per square inch sectional area, or so much per pound.

The only way is to consider all these relatively. To make such relative comparison all dimensions of bars, all details of procedure, and all machines, must be exactly similar, and all records must be the actual results of the tests of these bars.

This is what constitutes "Keep's Tests."

After the question of ultimate strength is decided, ability to withstand stress, without taking set, is the measure of usefulness for materials as ordinarily used.—To be fit for structural purposes, a material should be able to bear, without change of form, the stress to which it is subjected. To do this, it must be either perfectly rigid or perfectly elastic. In any construction, if any part remains out of shape after stress is removed, such distortion will either throw too much stress on some other part and thus imperil the structure, or (in such a case as that of an instrumental application, for example, of an engineer's transit), throw all parts out of adjustment.

In boiler-plates, extensibility before fracture is no doubt of great advantage. This is true of other materials under certain conditions. To determine what can be done with a metal, it is important to know all of its physical characteristics.

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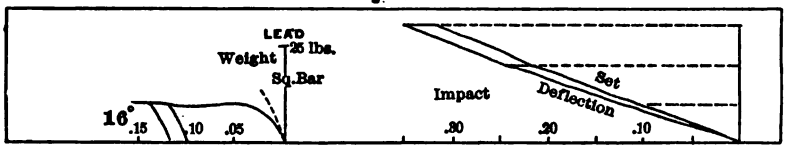
II.

Physical Tests of Metals and Alloys.

We intend, not to discuss generally the physical properties of the metals here considered, but only to present such new facts as our tests have revealed.

Generally, we have cast with our square test-bar a flat bar 1 inch wide and $\frac{1}{10}$ inch thick. Both bars are cast in moulds $12\frac{1}{8}$ inches long.

Fig. 8.

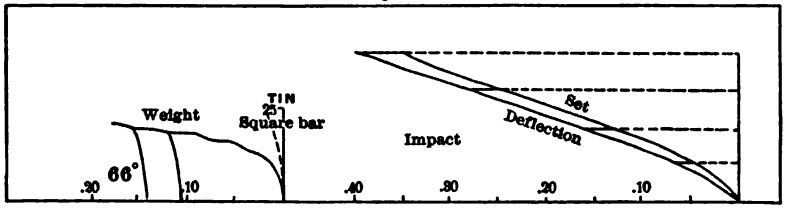


The Shrinkage is the difference in length between the mould and the bar when cold, and our records are expressed in hundredths of an inch of shrinkage per foot of length.

As a general thing, the square bar and the thin companion-bar do not shrink alike. The difference in shrinkage of these two bars exhibits many peculiarities of the metal. Generally, a greater shrinkage in the thin bar indicates that the cooling effect of the mould exerts a decided influence on the castings; in other words, that the metal has a tendency to chill, and *vice versa*.

In cast-iron, if the shrinkage of the thin bar is less than that of the square bar, the iron is not likely to take a chill, and *vice versa*.

Fig. 9.



Hardness is determined with one of Professor Turner's machines, and the record is the weight in grammes that causes a cutting diamond to make a standard scratch on a polished part of the test-bar.

Since the estimation of the standard scratch is, to some extent, a matter of judgment, too much importance should not be attached to a small difference in the recorded hardness of specimens.

It is safe, however, with such allowance, to compare the results

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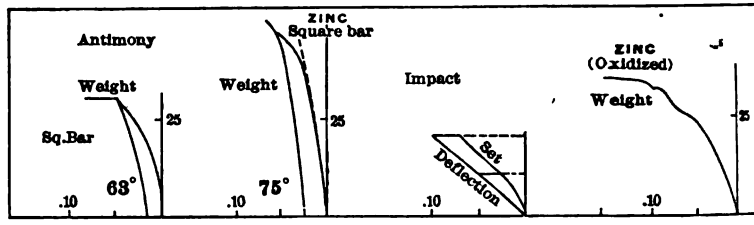
obtained by one observer. Hence, the records of hardness published in our various papers can be compared with reasonable confidence.

Lead.—Shrinkage, for the square bar, 0.113, for the flat bar, 0.101; hardness, 1 or 1.5.

As soon as the test-bar receives stress, the metal flows so rapidly that it is difficult to determine many of its characteristics. No

Fig. 10.



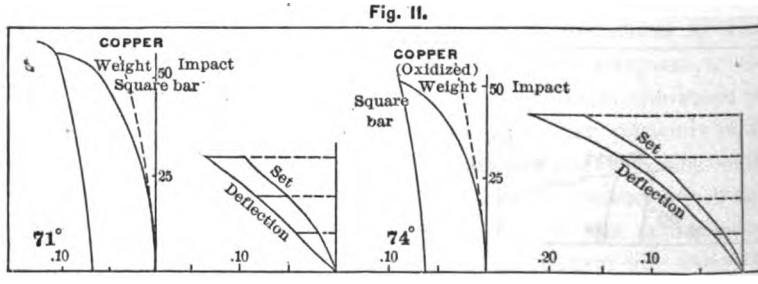
proportionality. Ultimate strength may be estimated at 10 pounds; total deflection before the bar wholly gives away, about 0.04; elasticity, 0.02; set from flow, 0.02; rigidity, 16°. The impact-test reveals nothing.

The spring-line is considerably curved.

This sample of lead was taken from tea-chests, and melted but once, and is therefore as soft and pure as commercial lead will generally be found to be.

Tin.—This metal, like lead, shows a strong tendency to flow. In such diagrams, the appearance of proportionality at the beginning of the diagram must be set aside.

Fig. 11.



Shrinkage, for the square bar, 0.053, for the flat bar, 0.051; hardness, 2; ultimate strength, about 20 pounds; total deflection, about 0.09 (due almost wholly to flow, and therefore not definite, as more or less time would have changed it); elasticity 0.015; rigidity, 66°. The impact-test shows nothing of interest.

The spring-line is considerably curved.

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Antimony.—This metal solidifies in flat crystals, with the cleavage across the bar, which causes the metal to be very weak under transverse stress.

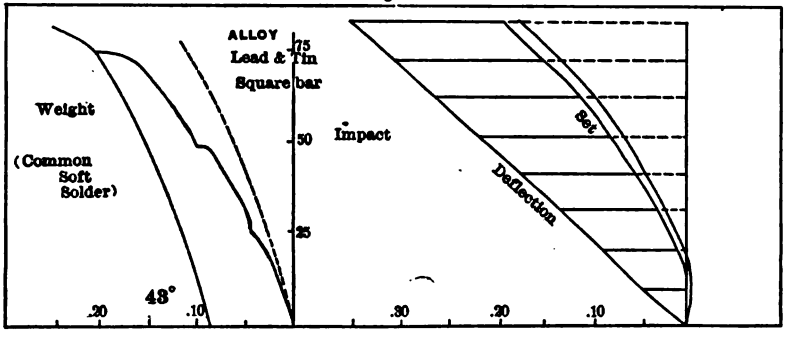
Shrinkage, for the square bar, 0.0.68, for the flat bar, 0.070; hardness, 16; ultimate strength, 31 pounds; proportionality ceases at about 12 pounds; total deflection, 0.045 (of which 0.03 is elasticity and 0.015 permanent set); rigidity, 63°.

We did not test by impact.

In grinding on an emery-wheel it heated very little. This property may be one of the reasons for its being introduced into most anti-friction metallic compounds.

Zinc.—Much depends upon the handling of zinc in melting, and much upon its purity. We show diagrams of two casts. One made

Fig. 12.



by an expert brass-founder, care being taken to prevent oxidation; and the second, from another lot of metal, melted in contact with the air. The crystallization of the latter is much coarser.

The shrinkage of the square unoxidized bar is 0.148; of the flat, 0.164; hardness, 6; ultimate strength, about 50 lbs.; proportionality ceases at about 30 lbs.; total deflection, 0.055; elasticity, 0.03; permanent set, 0.025; rigidity, 75°.

The impact-test does not possess any interest, nor does the diagram from the oxidized metal. The shrinkage of the latter was 0.189 for the square, and 0.165 for the flat bar.

Copper.—Copper is a difficult metal to form into castings from a peculiar tendency to swell after the mould is full, and from its rapid oxidation when in a molten condition. We procured the best commercial lake copper, and had our test-bars made by a brass-founder with the greatest care. The diagram, Fig. 11, shows that the metal flows as soon as stress is applied. Shrinkage, for the square bar,

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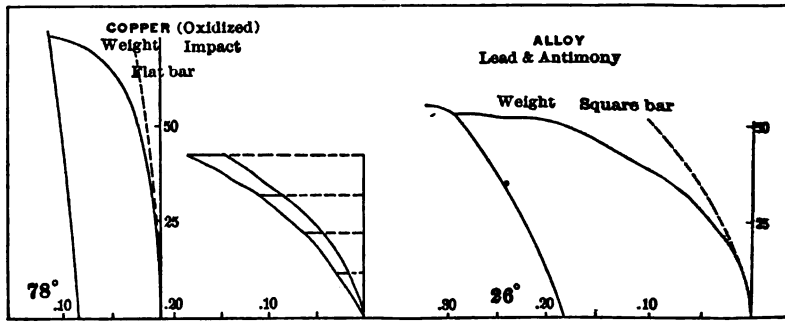
0.248, for the flat, 0.223; hardness, 12; ultimate strength, about 60 lbs.; no proportionality; total deflection, 0.105; elasticity, 0.04; permanent set, 0.065; rigidity, 71°. The spring-line is a curve.

A cast, which we made with no care to prevent oxidation, gave shrinkage for square bars, 0.23; hardness, 14; ultimate strength, 50 lbs.; elasticity, 0.03; set, 0.06; deflection, 0.09; rigidity, 74°. Mr. H. A. Tucker found the tensile strength of this last metal to be 9154 lbs. per square inch, with elongation 4 per cent. in 1 inch. In Fig. 13 we give diagrams of this metal from bars $\frac{1}{4}$ x 1 x 6 inches in size.

Alloys of the Preceding Metals.

Lead and Tin.—These were taken in equal parts by weight, forming common soft solder. This alloy has a lower melting-point than

Fig. 13.



either of the metals, and, like many other alloys, differs in every respect from its constituents. Lead has a shrinkage of 0.113, and tin of 0.053; but the alloy has 0.047 for the square, and 0.043 for the flat bar. It has twice the combined strength of both. Its hardness is 2; ultimate strength, 75 lbs.; no proportionality; total deflection, 0.18; elasticity, 0.11; set, 0.07; rigidity, 43°. The impact-diagram possesses considerable interest, especially as the bar takes no set with $\frac{1}{8}$-inch fall, and shows considerable vibration.

Lead and Antimony.—The latter metal is often used to harden lead. About 5 per cent. of it is used in this alloy.

It has a shrinkage of 0.064 for the square bar, and 0.063 for the flat, being less than that of either of the metals used. This is remarkable, as showing the small percentage of antimony necessary to take the shrinkage out of lead. Its hardness is 4, and its strength about twice that of antimony and six times that of lead, ultimate

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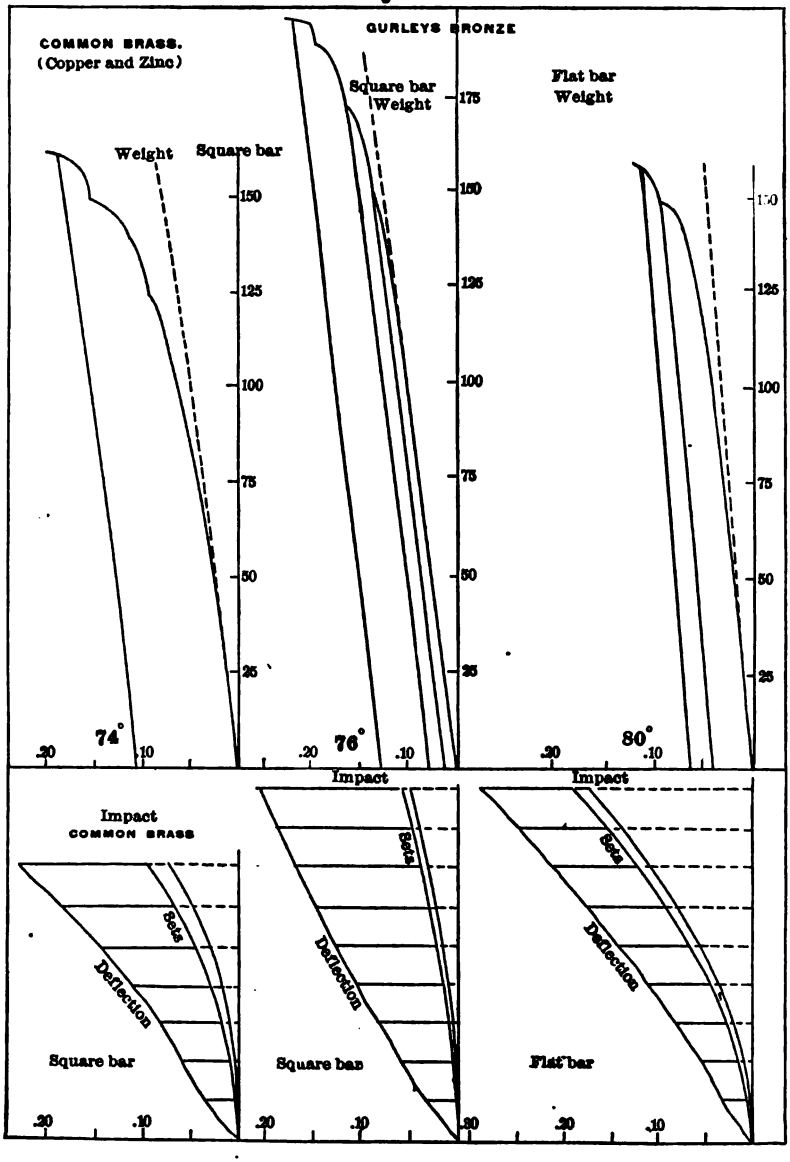
strength being 50 pounds. The diagram starts off in a peculiar manner, and yet we should wish to make other tests before stating

Fig. 14.



that proportionality did not cease until 20 pounds. Deflection is about 0.18; elasticity, 0.09; set, 0.09; rigidity, 26°.

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Copper and Zinc.—Five parts of copper and 4 parts of zinc. This is common brass, which is a very indefinite term, on account of the varying proportions necessary to produce a given quality of brass due to the varying quality of each metal. In compounding brass it is sought to use the greatest possible quantity of zinc, so as to cheapen the alloy.

Often, by using a brand of zinc costing a cent more per pound, it is possible to use very much less copper and not injure the quality of the brass.

In this case we used the same copper and zinc as that from which we have shown tests. The diagrams in Fig. 14 show as remarkable results as any of the preceding alloys. The strength is three times that of either of the combined metals. The hardness is 10; the shrinkage, about the mean between that of copper and zinc, being 0.190 for the square, and 0.191 for the flat bar; ultimate strength, about 150 lbs.; proportionality, about 35; deflection, 0.15; elasticity, 0.08; set, 0.07; rigidity, 74°.

Gurley's Bronze.—This is composed of 16 parts of copper, 1 of tin, 1 of zinc and ½ of lead. It is used by Messrs. W. & L. E. Gurley, of Troy, N. Y., for the frame-work of their engineers' transit instruments.

The bars tested were cast from the same pot from which frames were being poured. This alloy is remarkable when compared with the tests of the metals of which it is composed and with the record from brass bars. The ultimate strength is 200 pounds; proportionality continues until above 110 pounds; deflection is 0.15; elasticity, 0.10; set, 0.05; rigidity, 76°; hardness, 24. The ductility of this alloy is very great. Under impact-test it does not receive set until the ¼-inch fall, which is more severe treatment than any surveying instrument is likely to receive. The shrinkage is low, being, for the square bar, 0.177. We append diagrams from a bar ¼ inch thick, 1 inch wide and 6 inches long.

We sent bars for tensile test to Mr. H. A. Tucker, who reports its tensile strength 41,114 pounds per square inch; elongation, 27 per cent. in 1 inch; sp. gr. 8.696.

III.

ALUMINUM.

Commercial aluminum contains several foreign substances, the most common being silicon and iron. It is a question to be solved in the near future to what uses this metal may be advantageously applied.

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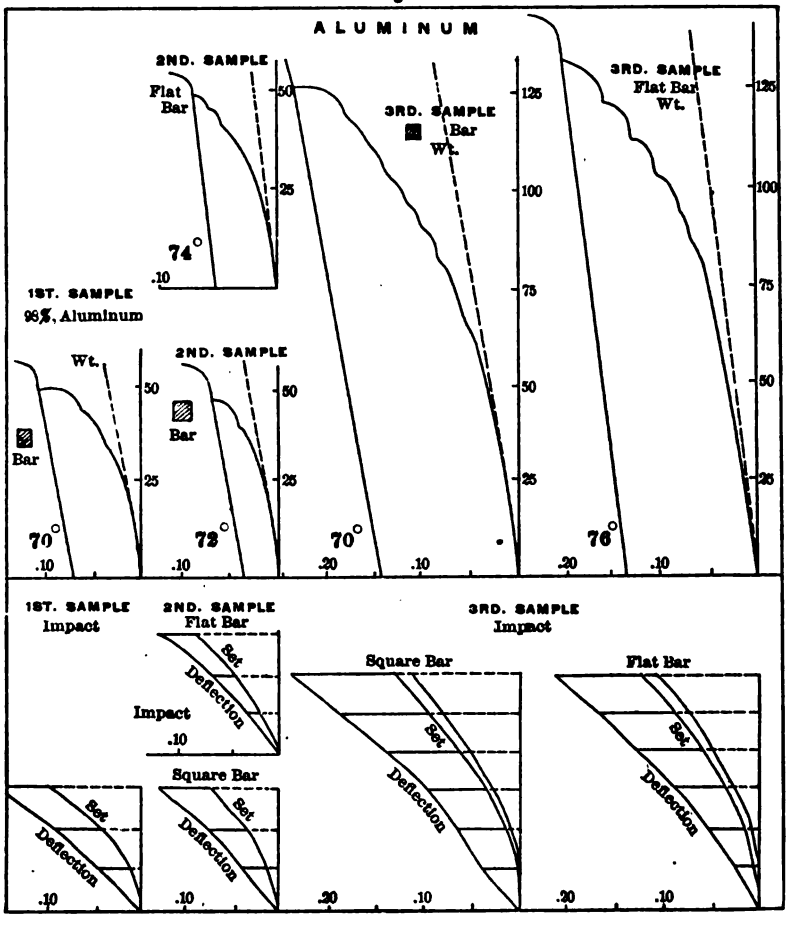
Owing to its lightness it seems desirable that instruments that are to be transported by hand should be constructed of this metal. With this idea in view we have presented in the foregoing pages the data regarding other metals, especially of the bronze used in the construction of surveying instruments.

An engineer's transit, with a 5-inch needle, which is a standard

Fig. 15.



instrument, weighs in bronze about 16 pounds complete, exclusive of box and tripod. If made of aluminum (all but screws, sockets, axis of telescope, etc., which must be of bronze) its weight would be reduced about 40 per cent., or to 9.6 pounds. An engineer carries his instrument anywhere from 5 to 25 miles per day, often over very rough country, and any decrease in weight would proportionately

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lessen his fatigue. The time is not far distant when the price of aluminum, compared bulk for bulk with other metals, will be as cheap as that of bronze. We wish to present tests of several grades of commercial aluminum and several aluminum alloys to indicate in which direction experiments should be made to meet the requirements of practical use.

Aluminum may be rolled to any thickness or drawn into rods and tubing. It can be pressed into almost any form, and most beautiful silver-white castings are easily produced from it.

It is in the latter shape that the bulk of any instrument must be made. It is therefore in the cast form that we are most interested. We shall present tests of several samples in the order of their purity.

First Sample.—*Al*, 98 per cent.; *Si*, 1.34; *Fe*, 0.32 per cent. This metal was made by the Pittsburgh Reduction Company, of Pittsburgh, Pa., by the Hall process.

In Fig. 15 we show, on the left hand, diagrams of both weight and impact. It shows proportionality for about 12 pounds. Its ultimate strength is 50 pounds. Total deflection, 0.08; elasticity, 0.035; set, 0.045; rigidity, 70°.

The shrinkage was 0.185 for the square bar, and 0.170 for the thin bar cast with it. This shows that aluminum will not take any chill in being cast in a mould. The relative hardness of this sample is 10. This is the hardness of commercially pure platinum, and as this sample is as pure as the metal will generally be found, we may adopt 10 as the hardness of aluminum. It is also the hardness of the brass tested in Fig. 14. In filing this sample, the teeth of the file filled a little less than with brass.

Second Sample.—*Al*, 95.50; *Si*, 1.62; *Fe*, 2.88 per cent. This is also from the Pittsburgh Company. Shrinkage, 0.191 for the square and 0.188 for the thin bar. The upper diagram is made from a flat bar cast with the above, $\frac{1}{2} \times 1 \times 6$ inches in size. The records for the square bar, as shown by the diagrams below this, are: ultimate strength, 50 pounds; proportional for about 12 pounds; deflection, 0.065; elasticity, 0.03; set, 0.035; rigidity, 72°; hardness, 12.

It will be noticed that with aluminum all spring lines are straight. Mr. Tucker found the sp. gr. to be 2.72. His results for tensile strength were:

No. of Specimen.	Tensile Strength. Pounds per sq. inch.	Elongation. Per cent. in 1 inch.	Sp. gr.
1	17,894	3	2.75
2	17,139	5	2.72
3	17,411	5	2.72
5	17,255	6	2.72

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The Pittsburgh Testing Laboratory found the following results for compression of samples sent them:

No. specimen.	Original dimensions.	Original area.	Crushing strength, lbs.	Crushing strength, lbs. per sq. in.	Length before crushing.	Length after crushing.
1	0.492	0.1901	7600	39,980	1.058	0.594
2	0.490	0.1886	6000	31,810	1.040	0.695
3	0.487	0.1863	6500	34,890	1.050	0.823
4	0.490	0.1886	7900	41,890	1.050	0.617
5	0.490	0.1886	7500	39,770	1.054	0.590

Third Sample.—*Al*, 93.50; *Si* 3.80; *Fe*, 2.50 per cent. This is also from Pittsburgh.

The shrinkage is 0.182 for the square and 0.168 for the thin bar. We give at the extreme right of Fig. 15 diagrams of both weight and impact from a flat bar $\frac{1}{4} \times 1 \times 6$ inches in size. At the central part of the figure we show diagrams from square bars. The ultimate strength is 125 pounds and proportionality continues to about 25 pounds. Deflection is 0.19; elasticity, 0.08; set, 0.11; rigidity, 70°; Hardness, 12.

Fourth Sample.—This aluminum was made under the Grätzel patents by the Aluminium Magnesium Fabrik Co., Bremen, and was furnished to us by Charles T. White & Son, 134 Pearl street, New York.

The Pittsburgh Testing Laboratory found this to contain *Al*, 93.20 *Si*, 5.95; *Fe*, 0.85.

The manufacturers claim that the metal probably contained *Al*, 96.96; *Si*, 0.59; *Fe*, 1.790.

We present a diagram of a square bar of this metal in Fig. 16, showing ultimate strength, about 100 pounds; proportionality continuing past 30 pounds; deflection, 0.16; elasticity, 0.07; set, 0.09; rigidity, 68°. The shrinkage of the square bar was 0.166.

Mr. Theodore Stevens tested samples of this metal for us with the following results:

No. of Specimen.	Tenacity in pounds per square inch.	Elongation. Per cent. in one inch.	Sp. gr.
1	19,380	1	2.66
2	19,060	2½	
3	18,395	2	
4	18,630	1	
Average,	18,866	1.6	

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Fifth Sample.—Al, 96.36; *Si*, 2.16; *Fe*, 1.47; *Cu*, 0.02.—This was a small cast sample $\frac{1}{4}\times1\times6$ inches in size, made by the Pittsburgh Reduction Co. and sent in a letter. It shows remarkable physical properties; we show a weight-diagram from this flat-bar in Fig. 16, and below it an impact-diagram. (To allow comparisons to be made we have recorded in Fig. 15 diagrams from the same sized flat-bars of samples 2 and 3.)

In this bar ultimate strength is about 120 pounds; proportionality extends to about 70 pounds; deflection is 0.08; elasticity is 0.05; set 0.03; rigidity, 78°; hardness, 10.

The impact-diagram brings out the good qualtities of this metal. It bears impact much better than the bronze in Fig. 14. It requires but little if any improvement to make it equal in every way to that metal. We sent a bar to Mr. Tucker, who reported that the bar broke through a large slag hole, but not until it had withstood 13,849 pounds per square inch tensile stress. The elongation in one inch for this stress was 5 per cent. Specific gravity was 2.67.

Increase of Strength and Rigidity of Aluminum.—Some method is needed for increasing these properties in this metal. When the metal is supplied in the shape of rolled bars, wire, sheets or tubes, aluminum is much stronger than when cast.

To exhibit the effect of rolling, we add in Fig. 16 diagrams from a rolled bar of Grätzel's aluminum of the same quality as the fourth sample. The bar was $\frac{1}{4}\times1\times6$ inches in size; ultimate strength is about 200 pounds; proportionality continuing to 120 pounds; deflection, 0.14; elasticity, 0.09; set, 0.05; rigidity, 78°.

We endeavored to cast a flat-bar in the same shape from this rolled piece, but by mistake the metal was spilled in the furnace and was recovered in the ash-pit. From its color it has evidently taken up some iron. The cast-bar is therefore not exactly an example of what a cast-bar of the same dimensions would show, though it makes a record that must be near the truth. A comparison of this bar with the rolled bar shows the amount of strength that the metal gained from rolling. The strength of the cast-bar is about 100 pounds; proportionality about 30; deflection, 11; elasticity, 0.055; set, 0.055; rigidity, 74°.

The strength which Barlow gives for aluminum is 26,800 pounds per square inch (tensile). He says he tested a bar cut from a rod (probably rolled) $\frac{1}{4}$ inch square and 3 feet long. This metal would probably be stronger than the Grätzel metal, that we have tested, for it was rolled on four sides, while the bar which we tested was cut from

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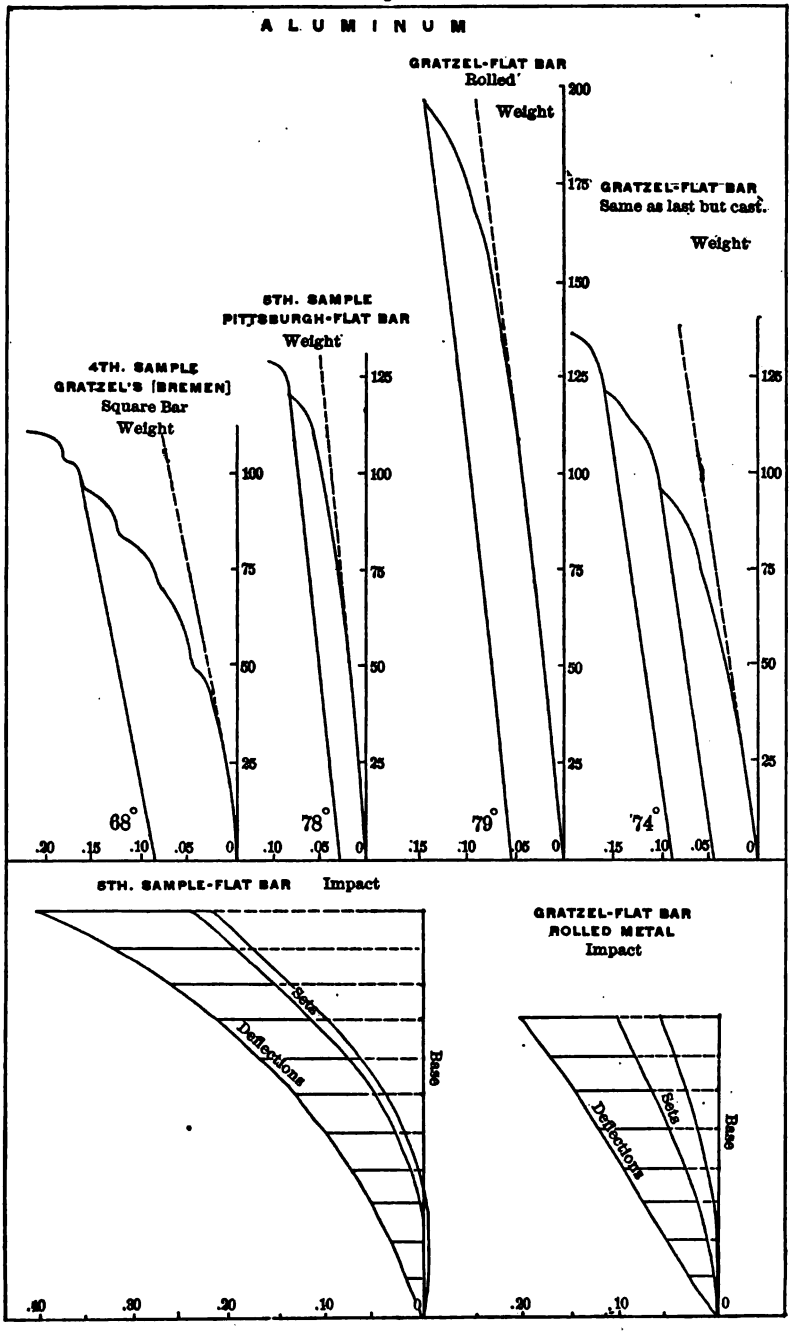
Fig. 16.



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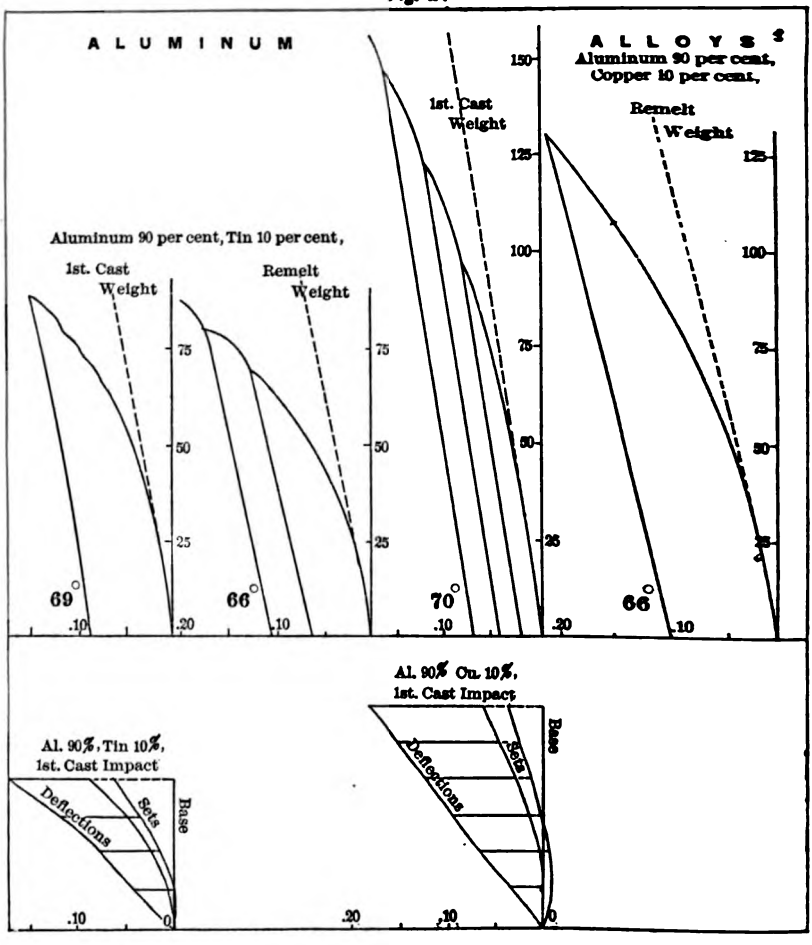
a sheet ¼ inch thick. If we compare the tensile strength, 18,866 of the cast bar of this last metal (fourth sample) with Barlow's strength, we find on comparing the diagrams of cast and rolled in Fig. 16 that the transverse strength varies in about the same ratio.

Fig. 17.



ALUMINUM ALLOYS.

We have made a few experiments to ascertain the effect produced by additions of copper and of tin to aluminum, the results of which are presented below.

Aluminum and Tin.—Al (sample 3), 90 parts, and tin (Fig. 9) 10 parts by weight. We show diagrams from this alloy in Fig. 17.

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Shrinkage, 0.157 for the square, and 0.152 for the thin bar (a great reduction for so small a percentage of tin); ultimate strength, about 85 pounds; proportionality, 25; deflection, 0.15; elasticity, 0.06; set, 0.09; rigidity, 69°; hardness, 8.

The alloy was remelted to see if this would improve its qualities. Its shrinkage was 0.148 for the square, and 0.142 for the thin bar; strength about 80 pounds; proportionality, 25; deflection, 0.17; elasticity, 0.06; set, 0.11; rigidity, 66°; hardness, 14.

The spring-line of aluminum seems to be straight, but the 10 per cent. of tin changes it to a curve. It is straighter in the remelt.

Aluminum and Copper.—We took 90 parts by weight of aluminum (Sample 3) and added 10 per cent. of the purest copper wire. The result was in every way beneficial, but not to any great extent. The shrinkage for the square bar was 0.141, which shows a remarkable decrease when we consider the very high shrinkage of copper, and also of aluminum. The diagram in Fig. 17 shows ultimate strength to be about 150 pounds; proportionality continuing to more than 50 pounds; deflection, 0.17; elasticity, 0.09; set, 0.08; rigidity, 70°; hardness, 10. The spring-line is straight. The alloy resists impact better than without the copper addition.

Aware of the impression that bronzes improve by being remelted, we cast this over again; but we found it to deteriorate in every way. Its shrinkage was now 0.137 for the square and 0.098 for the thin bar; strength, 125 pounds; proportionality, 30; deflection, 0.20; elasticity, 0.10; set, 0.10; rigidity, 66°; hardness, 18. The metal at the part of the bar lowest in the mould is white and crystalline, like aluminum, while the upper part of the casting is darker when viewed with the naked eye. This color is partially due to the metal being a mass of minute blow-holes. The inner surfaces of many of these holes are more or less iridescent, and between the honey-combed upper part and the solid lower part is a layer of blow-holes much larger than those above it.

From this result it seems that a small amount of copper does not alloy well with a large percentage of aluminum, yet this behavior may be due to some other peculiarity in the alloy that we have not noticed.

Aluminum Bronze.—This is composed of about 90 per cent. copper and 9 per cent. aluminum. This mixture behaves in a very different manner from the alloy last considered, in which the percentages are reversed.

The two samples of which we show diagrams were made in the

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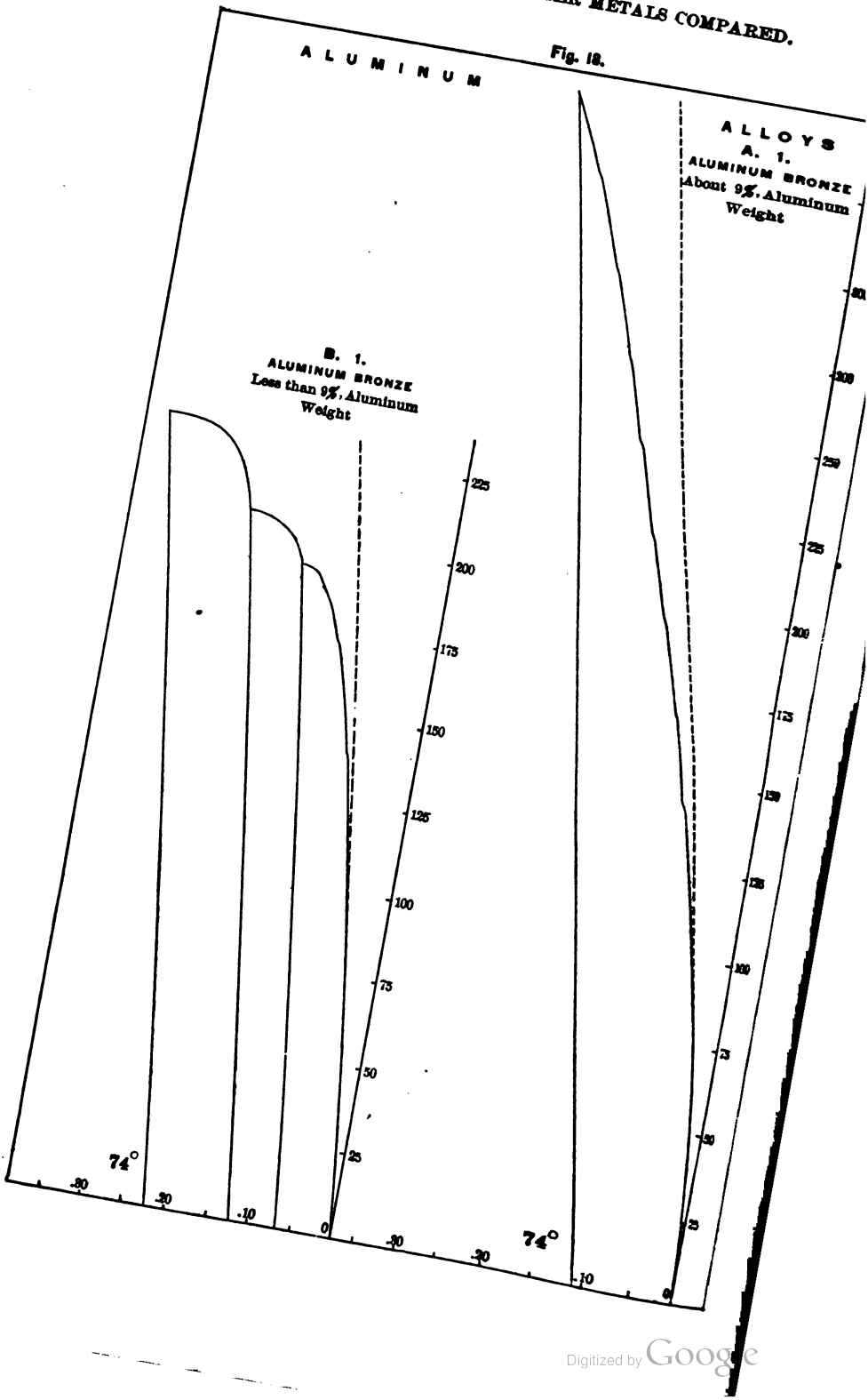
ALUMINUM AND OTHER METALS COMPARED.

Fig. 18.



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Cowles electric furnace. The B 1 metal contains less than 9 per cent. Al, and the A 1 metal is said to contain about 9 per cent., and between one and two per cent. of silicon.

The B 1 bronze has a shrinkage of 0.258 for the square, and 0.243 for the thin bar; strength, about 190 pounds; proportionality, 80; deflection, 0.15; elasticity, 0.105; set, 0.045; rigidity, 74°; hardness, 20.

	Name of metal or alloy.	Aluminum.	Silicon.	Iron.	Ultimate strength.	Limit of proportionality.	Deflection.	Elasticity.	Set.	Rigidity.	Hardness.	Shrinkage.
1	Cast-iron	.....	.....	.....	366	100	.21	.175	.03½	77	...	.....
2	Steel spring	.....	.....	.....	800+	800+	.18	.180	.00	83½	...	.....
3	Wrought-iron (rolled)	.....	.....	.....	445	430	.12½	.115	.01	83	...	.....
4	Wrought-iron " straight'd	.....	.....	.....	445	75	.21	.115	.09½	83	...	.....
5	Lead	.....	.....	.....	10	0	.04	.025	.02	16	1	.113
6	Tin	.....	.....	.....	20	0	.09	.015	.....	66	2	.053
7	Antimony	.....	.....	.....	31	12	.045	.03	.015	63	16	.068
8	Zinc	.....	.....	.....	50	30	.055	.03	.02½	74	6	.148
9	Copper	.....	.....	.....	60	0	.105	.04	.065	71	12	.248
10	Lead and tin	.....	.....	.....	75	0	.18	.11	.07	43	2	.047
11	Lead and antimony	.....	.....	.....	50	20	.18	.09	.09	26	4	.064
12	Brass	.....	.....	.....	150	35	.15	.03	.07	74	10	.190
13	Gurley's bronze	.....	.....	.....	200	110	.15	.10	.05	76	24	.177
14	Aluminum, 1st sam. (Pittsb'h)	98.00	1.84	.32	50	12	.08	.035	.045	70	10	.185
15	" 2d " "	95.50	1.62	2.88	50	12	.065	.03	.035	72	12	.191
16	" 5th " (Bremen)	96.36	2.16	1.47	120	70	.08	.05	.03	78	10	.....
17	" 3d " (Pittsb'h)	93.50	3.80	2.50	125	25	.19	.03	.11	70	12	.182
18	" 4th " "	93.20	5.95	.85	100	30	.16	.07	.09	68	...	.166
19	Al 90 per ct. and tin 10 per ct.	.....	.....	.....	85	25	.15	.06	.09	69	8	.157
20	The same remelted	.....	.....	.....	80	25	.17	.06	.11	66	14	.148
21	Al 90 per ct. and copper 10 per ct.	.....	.....	.....	150	50	.17	.09	.08	70	10	.141
22	The same remelted	.....	.....	.....	125	30	.20	.10	.10	66	18	.137
23	Aluminum bronze, B1 (Cowles)	.....	.....	.....	190	80	.39	.105	.045	74	20	.258
24	" " A1 "	.....	.....	.....	400	90	.15	.145	.145	74	20	.246
25	Maplewood	.....	.....	.....	124	121	.80	.80	0	28	...	.....

The A 1 bronze has a shrinkage of 0.246 for the square, and 0.242 for the thin bar; strength, about 400 pounds; proportionality continuing above 90 pounds; deflection at 325 pounds is 0.29; elasticity, 0.20; set, 0.09; rigidity, 74°; hardness, 20.

At 400 the curve of the diagram had not materially changed. The deflection was 0.39; elasticity, 0.145; set, 0.145. The spring-line of aluminum bronze is a curve.

The impact-diagrams bring out the peculiarities of the metal.

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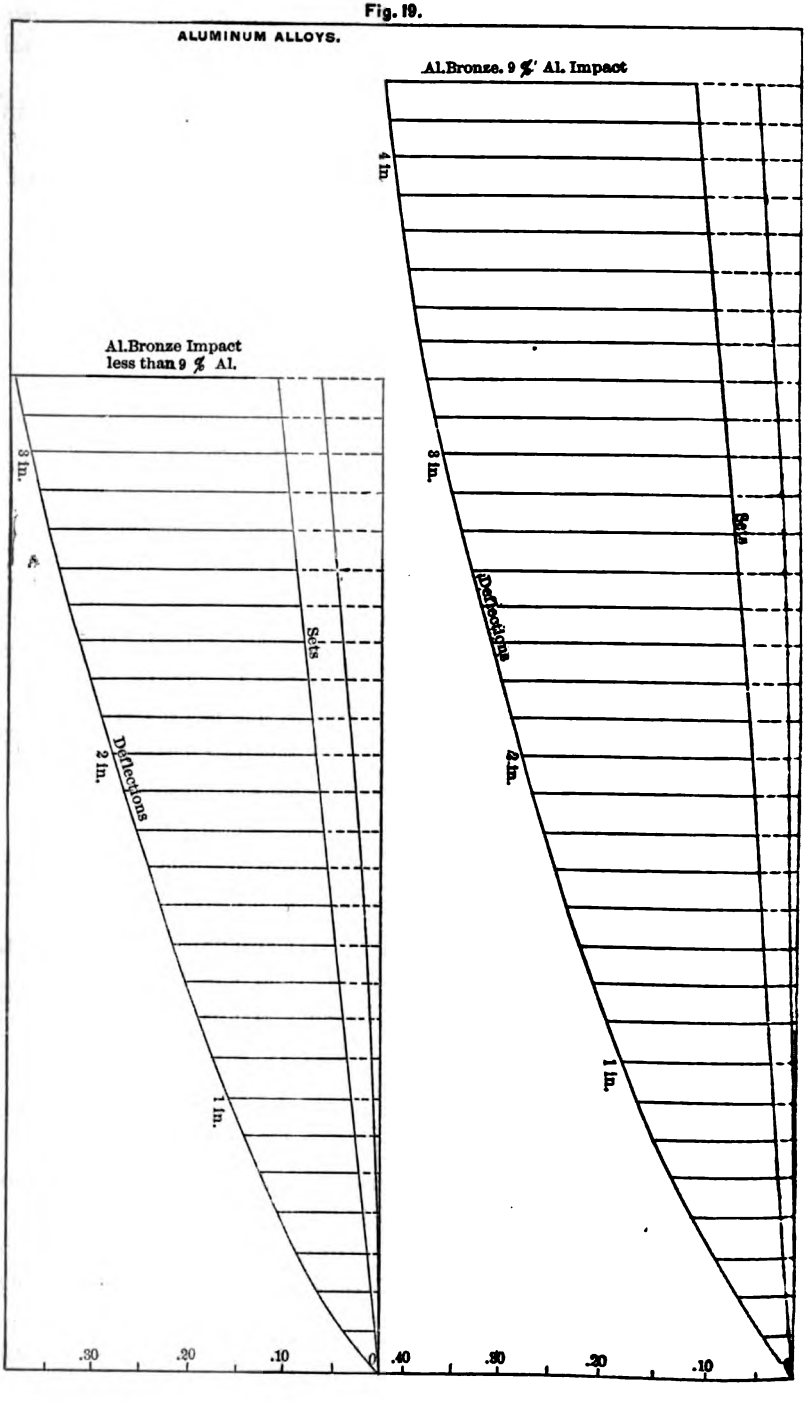
Fig. 19.



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These diagrams were taken from the same bars that were first tested by weight, and are not, therefore, as interesting as they would be, if we had used a different bar. One reason why we have not referred to the impact-diagrams of this paper more often, is that often the same bar has been used for both tests.

The Effect of Silicon and Iron on Aluminum.—To allow of a comparison of the samples of aluminum which we have tested, we have brought the results together in tabular form.

The elements which should be especially considered, are strength, proportionality, elasticity and rigidity. The higher these are, the more favorable. The silicon increases quite regularly from the first to the last, and, as a general thing, all of the above-named physical results show a favorable influence for silicon. It would seem from the above, and from other results, that silicon is the natural alloy for aluminum.

If this shall prove to be the case, it will be a most fortunate thing, for silicon occurs in all aluminous ores, and it is difficult to prevent its presence in the metal. Again, silicon is lighter than aluminum, its specific gravity being only 2.34. It is said that the presence of silicon causés aluminum to oxidize, but among the above-mentioned samples we have not found any more tendency for one to oxidize than another.

It is still desirable that some material shall be found that shall harden aluminum and add even more strength than is given to it by silicon.

If iron, in very small quantities, has any effect upon the metal, it hardens it. It increases the size of the crystals, and when present in quantity as great as 5 per cent., the crystals have flat bright facets about $\frac{1}{8}$ inch across, and the metal has become of no value.

Reports of Great Strength for Aluminum.—The question may be asked: Why do we so often read of the great strength of aluminum?

In his book on Aluminum, Richards quotes (p. 63) from a paper read by W. H. Barlow, before the British A. A. S., in 1882, that the tenacity of steel is 78,000 and of aluminum 26,800 pounds per square inch, and says:

"It thus appears that, taking the strength of aluminum in relation to its weight, it possesses a mechanical value about equal to that of steel of thirty-five tons per square inch tensile strength."

This may have been where the idea of great strength originated.

The great advantage that aluminum is to have over other metals is its lightness; therefore, we should compare equal bulks and not

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equal weights. The misunderstanding here arises from the scientific meaning of "mechanical value" not being commonly understood.

To show the danger of misapprehension from such a method of computation, examine the diagrams from the bar of maple wood and from the bar of tempered steel, and reason in the same manner. Mr. Tucker found for us that the tensile strength of this stick of maple wood was 16,464 pounds per square inch (with no elongation).

One cubic foot of maple weighs 47 pounds. One cubic foot of steel weighs 490 pounds, with a tensile strength of 78,000 pounds per square inch. Put this into the form above quoted and we have:

In relation to weights, the mechanical value of maple wood is more than twice that of steel.

One of the reasons that steel is used for beams is its great strength when compared with wooden beams of equal *size*. When we begin to compare maple beams of equal *weight* with an iron beam it must be ten times as large. Other questions present themselves when we talk of using such ponderous wooden beams which do not admit of a fair comparison being made.

Aluminum is desirable on account of its lightness, and we hope an alloy may be made which shall give the requisite properties for forming castings which shall be suitable for all uses to which brass and bronze castings are now applied.

Discussion.

Prof. D. S. Jacobus, Hoboken, N. J. (Communication to the Secretary):—It has been shown in practice that transverse tests of cast-iron and similarly brittle material determine its qualities in a thoroughly reliable way, and as these tests are more readily made than those by tension they have been quite generally adopted by foundrymen who wish to compare the qualities of different mixtures. It would be a great step in advance if all the tests were conducted in the same way so that they might be readily compared. If one standard, or several standards, of forms and sizes for test-bars could be made use of by the parties conducting the tests, we might then make a direct comparison of the results obtained. Mr. Keep has constructed a machine that is convenient in its operation, and, in addition to this, records automatically the load placed on the bar and the distortion which the load produces. An automatic record of this character is always desirable, if it is reliable; and

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that given by Mr. Keep's machine should be entirely so when ordinary care is employed in making the tests. We therefore consider that if any standard method of conducting transverse tests were generally adopted it would be well to copy the features of the above tests, because they involve many good points that have been brought out by the practical nature of the work already done on Mr. Keep's machine. The machine for testing by impact may show characteristics not apparent when a dead load is used, and these characteristics may be coupled with some quality of the metal that will greatly affect its commercial use. We hope to hear from Mr. Keep in regard to any further experiments he may make on this machine.

ALUMINUM IN WROUGHT-IRON AND STEEL CASTINGS.

BY W. J. KEEP, DETROIT, MICH.

(ANALYTICAL DETERMINATIONS IN CHARGE OF PROF. C. F. MABERY, CLEVELAND, OHIO.)

(Washington Meeting, February, 1890.)

WE have shown in former papers the influence which aluminum exerts when added to cast-iron, and we have presented in another paper, accompanying this one, the physical properties of the metal aluminum and of some of its alloys. The present paper continues the investigation. The reader will find, in our paper on "Aluminum and Other Metals Compared,"* explanations not here repeated, which will render the diagrams in this paper intelligible. The scale of stresses is laid off on the ordinate which rises from the zero-point of the diagram. The scale of deflections is laid off on the abscissa which runs horizontally through this zero-point.

By applying a given amount of stress, then running back to zero, and again applying the same stress, we describe a *spring-line*, in which the deflections are proportional to the stresses. Generally the deflections exceed this proportion during the original application of stress; and the excess is the *permanent set.*

By laying off from the zero-point a line parallel to the spring-line, we shall generally find that this new spring-line and the diagram coincide for some distance; that is, that the test-bar is

* *Trans.*, xviii., 798.

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generally a perfect spring for some amount of stress. The point where this spring-line and the original diagram begin to separate is the *limit of proportionality.*

The angle which the spring-line makes with the horizontal abscissa, when the scale of stress is 25 millimeters per 100 pounds, and the scale of deflections multiplies actual deflections five times, is called the *angle,* or *measure,* of *rigidity.*

The distance from the ordinate or *base-line* to the plotted spring-line is the *measure of elasticity* for any given stress. If measured on an abscissa perpendicular to the point marking the ultimate strength of the test-bar, this will be the *elasticity of the bar.*

The distance from the plotted spring-line to the original diagram *measures the set* of the bar for any given stress.

The *hardness* is the weight in grammes, which causes a cutting diamond to produce a standard scratch upon a polished portion of the fractured surface of the bar.

The test-bar may take *set* in either of two ways: by a change in the shape of the molecules of the bar, preventing the bar from returning to its original form when the stress is removed; or by a sliding of the molecules upon each other, which is called *flow.* Metals exhibiting it are said to be partially viscous or plastic.

We record both phenomena, as *set,* to avoid complication.

Taking Fig. 1 as an example: On the extreme left lower corner we give a diagram from our weight-machine of rolled wrought-iron. Next to this is an impact-diagram from another bar of the same metal. Next, one by weight and another by impact of bars of rolled ingot-iron.

The weight-diagram of wrought-iron shows the ultimate strength of 440 pounds; limit of proportionality, 340 pounds; deflection, 0.12; elasticity, 0.10; set, 0.02; rigidity, 83°; and hardness, 30. The diagram of rolled ingot-iron shows ultimate strength, 730 pounds; proportionality, 650 pounds; deflection, 0.20; elasticity, 0.16; set, 0.04; rigidity, 84°; and hardness, 38.

The impact-diagrams show total deflection, elasticity, set and strength, also the vibration. We have elsewhere called attention to the increased elasticity of most materials when subjected to impact. Each abscissa is the diagram for a blow having a fall $\frac{1}{8}$-inch greater than the blow next preceding.

We tested a $\frac{1}{2}$-inch square bar of the best tool-steel, with the temper drawn to a dark blue. This being a perfect spring, took no set. Its rigidity was 84°, the same as that of the ingot-iron;

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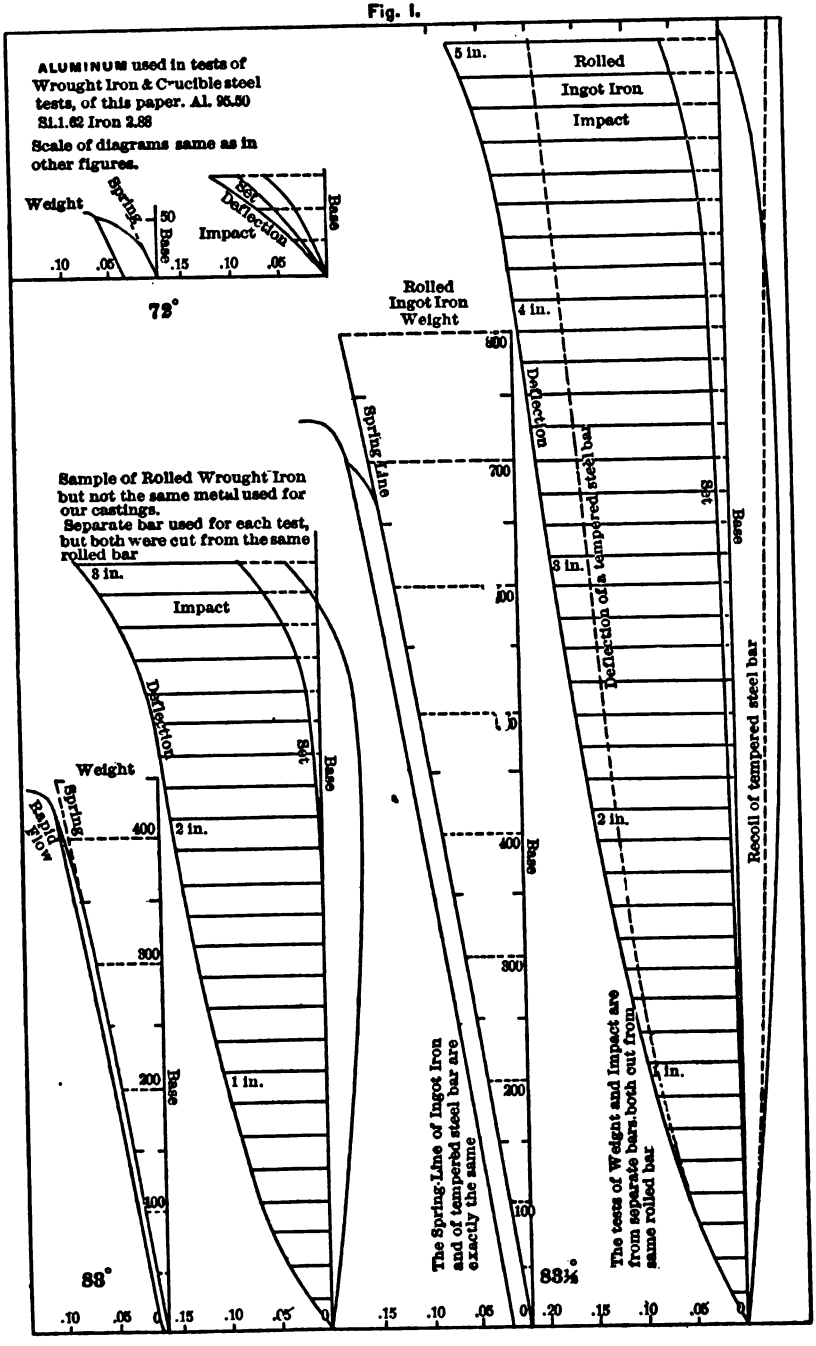
Fig. I.



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and we, therefore, for convenience, use the spring-line of that diagram, prolonged, as the diagram of this steel spring. We tested it to 800 pounds.

We then made this steel bar as hard as possible, when it broke at 400 pounds; but it had the same rigidity-angle of 84°, proving that temper does not increase rigidity.

The impact-diagram of this steel spring is shown by the dotted lines, laid over the impact-diagram of the ingot-iron in Fig. 1. As it takes no set, the diagram of set lies on the base-line ordinate.

Many interesting questions arising from the study of these diagrams will be discussed in a subsequent paper. We will here refer to a single one. As already observed, the spring-lines of the ingot-iron and of the steel-spring were exactly the same. For any stress less than 600 pounds, with the weight-test, elasticity and deflection are the same. According to authorities, the impact-stress should produce a like diagram for both. This is, however, not the case. The diagrams do not correspond in any respect, after the first half-dozen blows.

Those who have believed that the effect of impact can be correctly calculated from data obtained by a weight-test will be interested in these results.

At the top of Fig. 1 we give diagrams, reduced to the same scale as the other diagrams of this paper, of the aluminum used in the tests to be considered in this paper.

Aluminum in Remelted Wrought-Iron.

Wrought-iron, being manufactured without complete fusion is made up of fibers, interspersed with more or less slag, which has not been squeezed, forged or rolled out. Ingot-iron, being manufactured by fusion, is practically free from slag. After wrought-iron has been melted, the slag should have separated from the metal; and the product, which we in this paper call remelted wrought-iron, should not differ from ingot-iron, if the slag was the only difference in composition to begin with.

We are not aware that any record of tests has been published, showing the effect of aluminum in such remelted wrought-iron or ingot-iron.

Those who have seen the Mitis wrought-iron castings made in Sweden, or during the year 1885 at the foundry of the Worcester Malleable Iron Company, can feel only admiration of their beauty

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and perfection. In the patents for the process, it is declared that an exceedingly small percentage of aluminum added at a certain time, under certain conditions, will produce such castings.

In a small circular regarding Worcester Mitis, the tensile strength and elongation are given; and in another circular, published in 1885, analyses of Mitis, by Dr. Edward Riley, show as an average of seven samples: carbon, 0.099; silicon, 0.100; phosphorus, 0.153; and manganese, 0.022 per cent. No aluminum was found in this metal. It will be noticed that the above average of elements is about the same as the analysis of ordinary wrought-iron with the slag removed. The silicon and manganese are a little high. If the castings in Sweden, England and Worcester had been made with additions of silicon and manganese, or silicon alone, the extremely high temperature of the molten metal would have volatilized or oxidized most of the additions, so that the resultant metal should have had a composition similar to the above average.

It is stated by the owners of the Mitis patents that by remelting, the same result cannot be obtained, owing to the escape of the elements which prevented blow-holes in the first melting.

If aluminum had been introduced it probably would not have escaped and would have prevented blow-holes when the metal was remelted.

If the Worcester Mitis had been produced by additions of the high grade ferro-silicon which it is said that they used in their works for some purpose, it seems probable that with the skillful management that controlled the operation, just such beautiful castings could have been made.

Just about this time (1885) the Cowles alloy of aluminum and iron came upon the market. This consisted of carbon about 2.01; silicon, 3.86; and aluminum about 10 per cent.; the remainder being iron. We have, in a former paper, proved that aluminum and silicon work together in the same direction; and this alloy was especially adapted to the making of castings from melted wrought-iron.

We were anxious to obtain Mitis castings in the shape of our test-bars, and applied to every one that we could hear of who claimed to be working under these patents. But for some reason we could not obtain castings made from our patterns. A letter from one large company, dated March, 1889, says, "The process so far has not yielded entirely satisfactory results, when the attempt has been made to melt a very low-carbon wrought-iron."

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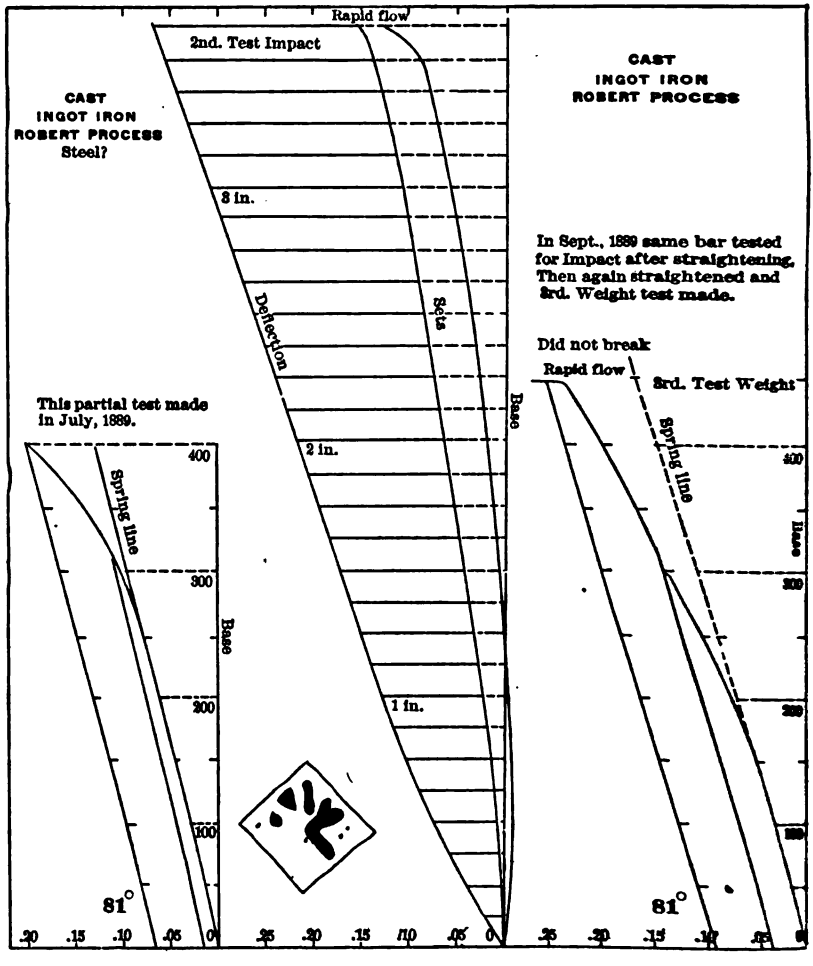
About this time we chanced to meet one of the agents of the Mitis Company, who agreed to have the castings made as we wished.

When told that we would furnish him all of the "J. B." wrought-iron and pure aluminum that he wanted for the work, he said that he should not know how to go to work with pure aluminum, and

Fig. 2.



asked to be permitted to use the Cowles alloy, as *they needed the other elements which that alloy contained.* He said that if he were to try pure aluminum he would begin with 0.25 per cent. and see what that would do. All this showed plainly that they considered an addition of carbon and silicon desirable.

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We therefore conclude that castings of wrought-iron and aluminum have not been made, or at least that no account of tests of such castings has been published. We shall in this paper present positive proof that very small percentages of aluminum will accomplish wonders, and also that it does not fail to act in remelting.

Small Quantities of Aluminum in Wrought-iron will prevent Blow-holes and will make Stronger Castings.—Before examining other tests, we present the results of an experiment to prove that it is not the addition of a small quantity of aluminum just before pouring, but the presence of the aluminum in the iron, which produces the results. At our request the owners of the Robert process for making ingot-iron, made test-bars for us of this metal.

We remelted it and cast another set of bars the tests of which are recorded in the diagrams of Fig. 2. The total carbon was 0.35 and the graphitic 0.04 per cent. The stress applied was about 400 pounds. Proportionality extends to 250 pounds. Deflection is 0.20 and elasticity 0.125; leaving set at 0.075; rigidity is 81° and hardness 45. The shrinkage was .220. These castings are very good, but contain a number of blow-holes. We give in Fig. 2 a drawing of the fracture of the bar showing holes.

At the same time that the above castings were originally made a second ladle of iron was caught, into which was put a 3-ounce piece of aluminum such as is shown in Fig. 1, after which another set of test-bars was made.

Owing to a mistake in proportioning the gates, these bars (as well as the first set) were cracked. We therefore remelted these and cast other test-bars from correctly made patterns. The results from the tests of these bars are shown in Fig. 3.

By calculation, the amount of aluminum in these castings was 0.20 per cent. Dr. Mabery found the total carbon to be 0.31 per cent. The stress applied to the remelted castings was 600 pounds; proportionality, 400 pounds; the deflection 0.15; the elasticity 0.13; the set 0.03; the rigidity 84°; and the hardness, 45, the same as without aluminum. The shrinkage was, as before, 0.220. The castings were perfectly solid. In the first cast, the grain of the metal alone was very coarse, but the cast containing aluminum was very fine and homogeneous.

The tests by impact were made on the same bars, after being straightened, which had been tested by weight (see Figs. 2 and 3); and we have shown in a former paper that a bar which has been bent and straightened will not, when again tested, give the same

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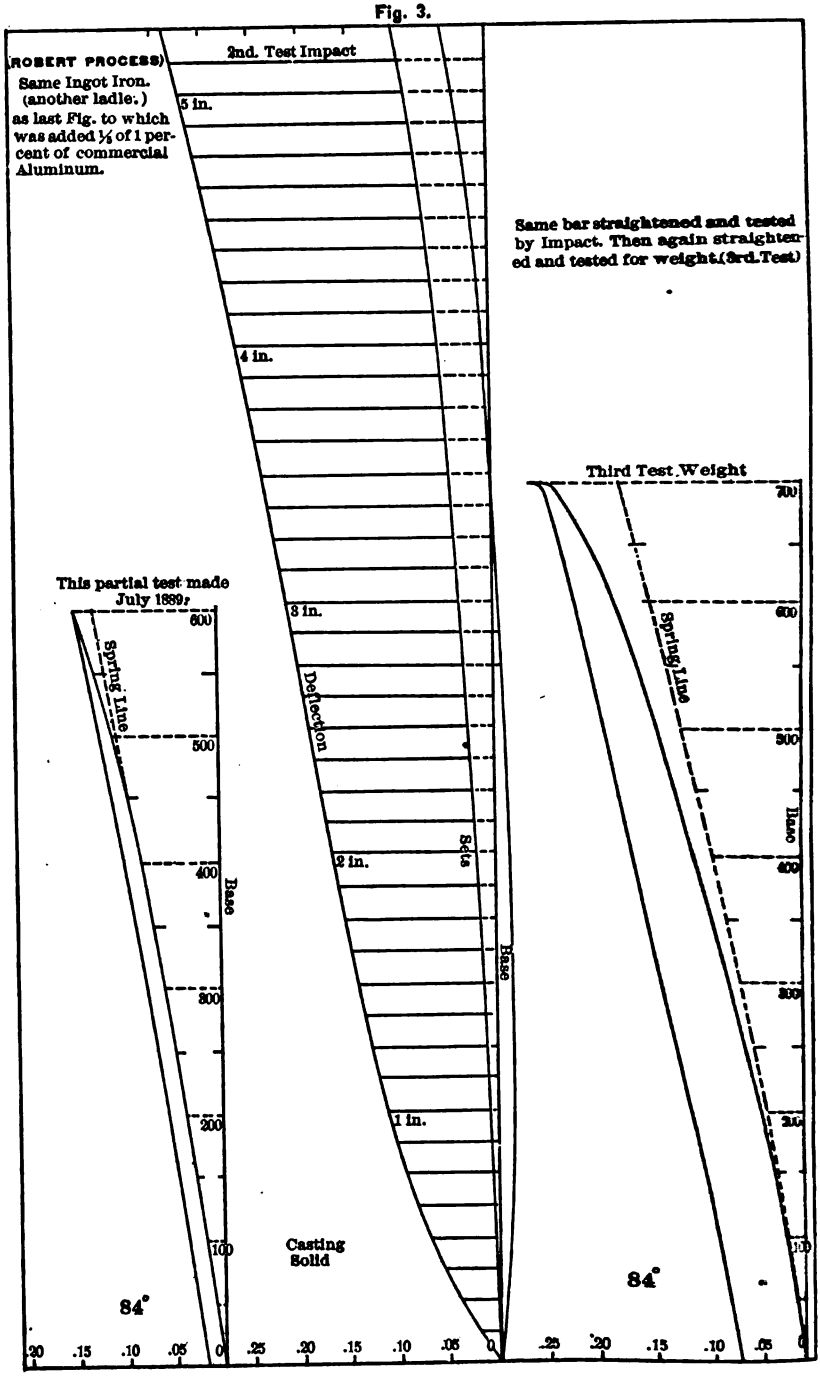
Fig. 3.



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results as an original test. It is interesting in both tests by impact to note the great deflection and elasticity without the bar giving way, as compared with the elasticity of the test by weight.

The same bars after impact-test were straightened and again tested by weight to ascertain the actual ultimate strength. The ultimate strength of the Robert ingot-iron was 450; proportionality, 125; deflection, 0.23; elasticity, 0.16; set, 0.07; rigidity, 81°. The rigidity is the same as in the first test. The elasticity at 400 seems to be greater.

The ultimate strength at the second weight-test (Fig. 3) of this same ingot-iron, but containing 0.20 per cent. of aluminum and total carbon reduced from 0.35 to 0.31, was 700 pounds. If at all proportional it is so only to 50 pounds. The deflection is 0.24; elasticity, 0.17; set, 0.07; and rigidity, 84°.

Both these sets of tests prove how bending and straightening deteriorate the good qualities of a metal.

What has the Aluminum done?—It has made the grain close and uniform. It has removed all blow-holes. It has raised the ultimate strength 55 per cent., and the limit of proportionality from 250 to 400. Elasticity at the limit of proportionality is the same in each. Rigidity is raised from 81° to 84°. Viscosity or tendency to flow is greatly diminished by aluminum. Power to resist shock is almost doubled.

As any ingot-iron can be taken directly from the converter and poured into the moulds, this experiment shows what a wonderful influence aluminum is destined to exert on the steel casting business, and the fact that such a casting can be remelted and made into perfectly solid castings shows that the aluminum remains in the metal and that its effect is lasting.

The tests which we shall next describe will, together with those which we have already presented, prove that an exceedingly high melting-temperature is necessary for the production of the best castings. The Detroit Steel and Spring Works, through their manager, Mr. Alexander DeLano, after hearing of the tests which we desired to have made, allowed their superintendent of melting, Mr. J. W. Allison, to make the casts for us. We prepared the moulds of graphite, and he placed in a plumbago crucible eighty pounds of "J. B." wrought-iron, which was supposed to contain about 0.087 of carbon and 0.056 of silicon.

We placed upon the iron platform near the furnaces containing the steel pots moulds marked 1, 2, 3, 4 and 5.

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It was doubtful whether we would be able to fill the mould No. 1 with melted wrought-iron, and much less any of the others.

We had attached to the ends of four long sticks of carbon, each one inch in diameter, pieces of aluminum. A three-ounce piece for cast No. 2; a seven-ounce piece for cast No. 3; five ounces for cast No. 4, and three ounces for cast No. 5. If we could make each mould take 13 pounds of iron these additions would make cast No. 1 contain no aluminum; cast No. 2, 0.25 per cent.; cast No. 3, 1.00 per cent.; cast No. 4, 2.00 per cent., and cast No. 5, 3.00 per cent. of aluminum.

The moulds were filled as follows: After the wrought-iron had been in the furnace six hours it was found to be melted and was withdrawn. The first mould was filled, leaving 67 pounds of iron in the pot. This was set down on the iron floor and the first stick of carbon was handed to the melter, who stirred in the three-ounce piece of aluminum and then poured the second mould. This should have left 54 pounds of iron in the pot. The character of the metal had not materially changed; it had not lost any of its fluidity. The pot was again set down, the seven-ounce piece of aluminum was stirred in and the third mould poured. The metal had become whiter and more fluid. There should have been 41 pounds of iron in the crucible, and this was again set down, the five-ounce piece was stirred in, the fourth mould was poured and the pot set down, when there should have been 28 pounds left. This metal was hotter and more fluid than at any previous heat. After stirring in the remaining three-ounce piece the iron became still more fluid and more white. When the last mould, No. 5, was poured, 15 pounds was left which was poured out on the platform, running into a plate about half an inch thick.

At each stirring in of aluminum, not only was the aluminum melted, but the stick of carbon was heated white hot.

Eighteen ounces of aluminum were melted and four sticks of carbon were raised to white heat, yet the last 15 pounds poured was more fluid than the first 80 pounds, which would hardly run.

Before giving other data we will refer to the analyses of these first five casts in tabular form near the end of this paper. Owing to the uncertainty of determinations of small percentages of aluminum the result of the analyses of cast No. 2 is not given. Owing to the piping of the metal in the mould each cast did not contain 13 pounds, thus leaving more iron in the pot after each cast than we expected. This reduced the percentage of aluminum in each cast

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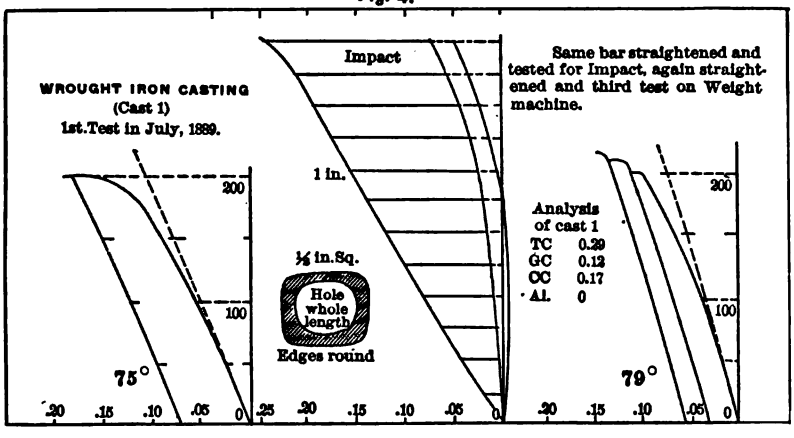
below what we had calculated. The reduced percentages are not, therefore, necessarily due to any loss of aluminum.

The casts contain more carbon than the J. B. iron was supposed to contain, and the metal no doubt took this from the crucible during the six hours that it was in the fire.

The carbons do not seem to be affected by the aluminum in any way.

Cast No. 1, *remelted wrought-iron*, contains, Al, none; total carbon, 0.29; graphitic, 0.12; combined, 0.17. The remelting entirely removes the fibrous character of the original metal, and, as the slag was all removed, the iron was changed to ingot-iron. The castings and runners were full of blow-holes, and the mould was not nearly

Fig. 4.



filled. We remelted this cast, and found it to melt much more easily than the "J B.," which we had not before been able to make fluid in our furnace. We cast a very good test-bar, from which we made the diagrams shown in Fig. 4.

The bar contained one blow-hole, extending from end to end, as shown. This imperfection caused the bar to show much less strength than if it had been solid; and this hollowness may have changed the other results.

The shrinkage of this bar was 0.292, which we may take as about the correct shrinkage of commercially pure wrought-iron. Its ultimate strength was about 200 pounds. It was proportional up to 50 pounds. Deflection, about 0.15; elasticity, 0.11; set, 0.04; rigidity, 75°; and hardness, 44.

The surface of the casting was very rough, being covered with

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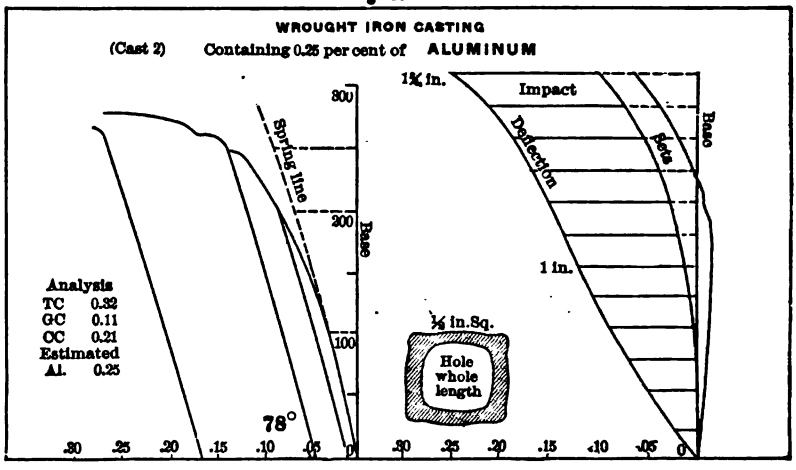
cavities caused by bubbles of gas. (All remelts are cast in ordinary green-sand moulds.)

The bar of Robert ingot-iron was nearly solid, and of nearly the same composition, so far as carbon is concerned, and its strength was 400.

Cast No. 2.—This contained nearly 0.25 per cent. of aluminum, by calculation, and analyzed total carbon, 0.32; graphitic, 0.11; combined, 0.21. The shrinkage was 0.281. The original casting was full of blow-holes, and did not run full. The remelt had a larger hole through its centre than the first cast. This does not prove that, with more heat, this percentage of aluminum would not have made a solid casting; for the Robert iron, with less aluminum, was perfectly solid.

Fig. 5.



The diagram, Fig. 5, shows that the ultimate strength is 250. There is no proportionality, on account of the hollow bar. Deflection, 0.13; elasticity, 0.09; set, 0.04; rigidity, 78°; hardness, 44.

The impact-test was made on a separate bar, and the bar bent in a flaw near the end, as well as at the center, which detracts from the value of the test.

Cast No. 3.—We intended that this should contain 1 per cent., but owing to too much metal remaining in the pot it contained only 0.58 per cent. aluminum, with total carbon, 0.33; graphitic, 0.085; combined, 0.245. This casting was practically sound. Its shrinkage is reduced to 0.273, although there seems no reason for its being so low. Fig. 6 shows that its ultimate strength is 430 pounds; it is

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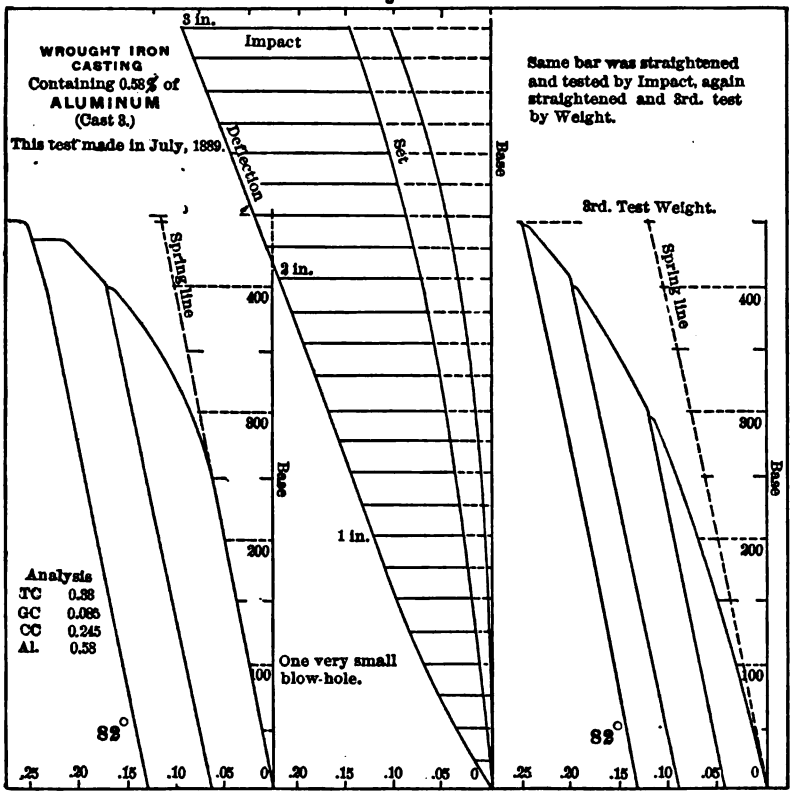
proportional to 250 pounds; deflection, 0.21; elasticity, 0.115; set, 0.095; rigidity, 82°; and hardness, 42.

If we had obtained a second bar for impact, it would not have taken set for several blows; but this bar, having been straightened, had lost its proportionality, as is shown by another weight-diagram, taken after impact for the purpose of showing the change in the character of the bar. This test gives the same ultimate strength

Fig. 6.



as the first, and no proportionality. Deflection and set are much greater, but elasticity, which is a quality due to the composition of the metal, is the same.

Cast No. 4 (see Fig. 7).—This should have contained 2 per cent., but actually had only 1.61 per cent. of aluminum. Total carbon, 0.37; graphitic, 0.12; combined, 0.25. This cast was solid, with a shrinkage of 0.282. This shrinkage is higher than it should have been. We were obliged to remelt this three times before we

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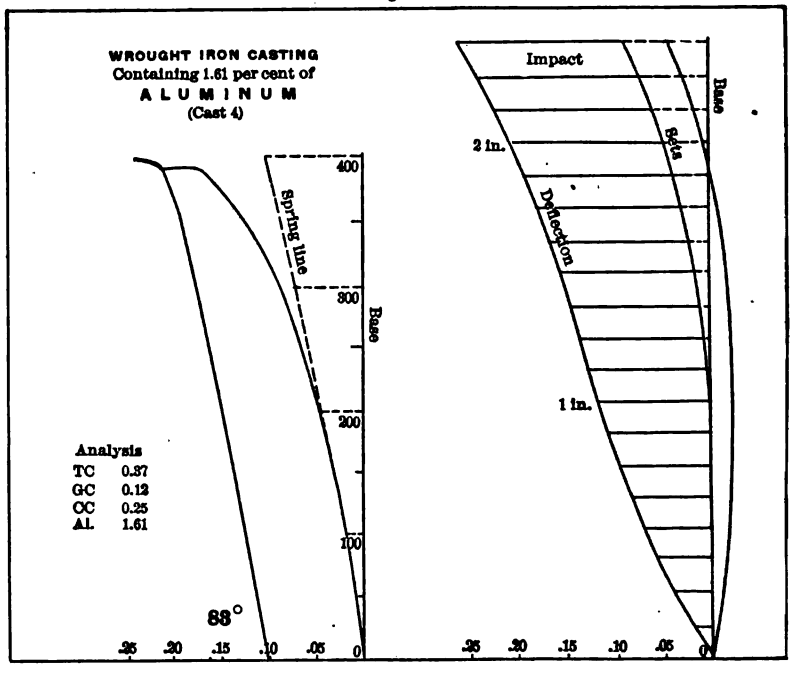
procured a good set of bars, and this may have influenced the shrinkage. All analyses were of the metal after the first mixture. The final castings have never been subjected to chemical analysis, it being thought that the physical tests prove conclusively that the aluminum is still present. Its ultimate strength is 390; proportionality, 250; deflection, 17; elasticity, 0.10; set, 0.07; rigidity, 83°; and hardness, 41.

In this case we had a second bar for impact-test, and may therefore make comparisons. The bar holds its proportionality until a

Fig. 7.



1-inch fall. Its elasticity at this blow is 0.115, while at the limit with the weight-machine it was only 0.06.

Cast No. 5.—In this the aluminum should have been 3 per cent., but was only 2.45. The casting of the remelt was smooth on the surface and perfectly solid. It melted almost as easily as white cast-iron. We succeeded in casting a thin companion-bar. The shrinkage of the square bar is reduced to 0.259, while the thin bar had a shrinkage of 0.290, showing that the rapid cooling increased shrinkage, but we could find no tendency to chill. The hardness is reduced to 30. This metal could be drawn out and welded, was bent

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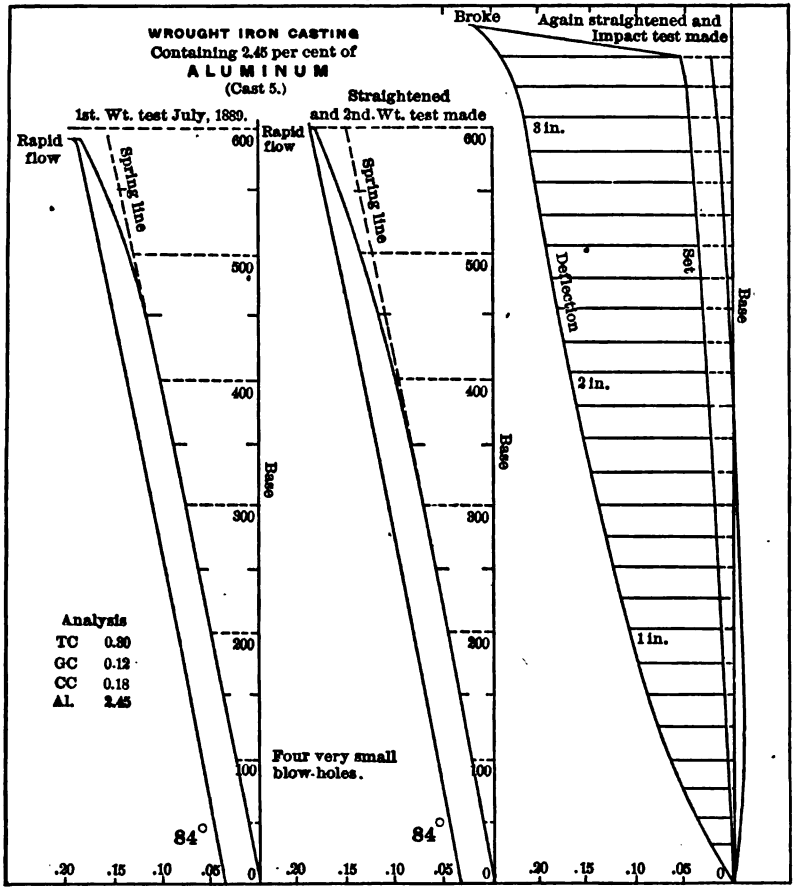
double when cold, and was in every way a good ingot-iron. The fracture of the bar exhibits a fine, compact crystalline grain, like good steel. The metal left in the pot after the mixture showed crystals with flat facets one-eighth of an inch across. We thus find that we have gone about as far in adding aluminum as it is wise to go. The fusibility of the alloy is low enough to allow it to be

Fig. 8.



easily melted. Its strength is higher than any iron we have tested, unless it has been strengthened by rolling. The bar shows other characteristics not shown by the previous casts.

In Fig. 8 we show diagrams from this metal. The first at the left is the original test, and gives ultimate strength, 590 pounds; proportionality, 450 pounds; deflection, 0.185; elasticity, 0.116; set,

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0.015; rigidity, 84°; hardness, 30. As we had only one bar, we straightened it and took another weight-diagram. The ultimate strength was the same as before, 600 pounds, and deflection, elasticity, set and rigidity are the same; but proportionality ran only to 300.

The bar was again straightened and tested by impact to fracture. As in other cases after straightening, this does not exhibit proportionality.

In this case we can compare ultimate strength, as the bar flowed so rapidly with weight at 600 that it could receive no more stress, and at a 3¼-inch fall the bar broke. The elasticity at this blow was 0.185, while at 600 pounds it was only 16. The deflection for the fall was 0.24, while for 600 pounds it was only 0.185.

Effects from Greater Percentages of Aluminum.—To ascertain what could be done in this direction, we began at the other end of the series, by adding 10 per cent. of the same wrought-iron we have just considered, to 90 per cent. of aluminum. The shrinkage was 0.161 for the square and 0.153 for the thin bar. It was not so strong as aluminum alone and was very brittle. The crystals had flat, shining facets and were very large.

We then made the proportion of iron 25 per cent. The shrinkage was 0.142 for the square and 0.127 for the thin bar. The crystals were mixed and the metal would hardly hold together.

We next made the proportions half-and-half. The bars broke by their own weight. The shrinkage of the square bars was 0.241, and the thin bars crumbled too much to be measured. These bars were left in a package for three months, when they had fallen to a fine powder.

This tendency to large cleavage which became prominent with 2.45 per cent. of aluminum and which proved so deleterious with larger percentages of aluminum, would indicate that for an iron and aluminum alloy the limit will be about 3 per cent of the latter, or perhaps a little higher.

We find, also, that with molten iron comparatively free from carbon, a very high temperature is necessary to enable a percentage of aluminum as small as one-quarter of 1 per cent. to make perfectly solid castings.

We do not consider our cast No. 2 as indicating the inability of this percentage of aluminum to make solid castings, because with the Robert iron, with the same percentage, the casting was solid. This shows that the manipulation, or some element in the compo-

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sition which we have not discovered, brought about a different result in the case of No. 2.

Aluminum in Steel Castings.

In former papers we have shown how varying percentages of aluminum influence carbon, where the iron is saturated with carbon, that is, in cast-iron.

We shall now show what aluminum will do where the iron is not saturated with carbon.

The preceding portion of this paper has treated of the influence of aluminum on iron containing about $\frac{1}{3}$ of 1 per cent. of carbon. Iron containing 1.5 per cent. of carbon would be a high-carbon steel, and could be forged and welded, but probably with difficulty. This percentage would be only one-half the carbon which would be contained if the iron was saturated with carbon.

Exactly the same course was pursued which was taken to get the aluminum into the wrought-iron, and a review of that description will apply equally well to this series of casts.

Enough carbon was added with the 80 pounds of "J. B." wrought-iron to bring the carbon in the steel to 1.50 per cent.

As in the former case, the moulds did not take as much metal as was expected, owing to piping; and the percentages of aluminum in each cast were therefore lower than was calculated. The casts were numbered 6, 7, 8, 9 and 10. After each addition of aluminum the metal grew more fluid and apparently hotter.

We here refer to the analytical determinations by Dr. Mabery, which are shown in the table on page 858.

Owing to the uncertainty of determinations of small percentages of aluminum, the result of the analyses of cast 7 and cast 2 is not given. Our tests prove its presence in these casts.

It will be noticed that aluminum expels carbon; but this is due to the expulsion of graphite already in the molten metal, and not to the change of combined into graphitic carbon. The combined carbon is even increased.

Owing to the influence of the 1.50 per cent. of carbon cast No. 6 ran full.

Cast No. 6.—As separate bars were tested for both weight and impact these records can be compared. As no manganese was added with the carbon the grain is very coarse. The diagrams are the two at the left of Fig. 9. The shrinkage is very irregular. In the two square bars made, when the original cast was remelted, it was 0.250

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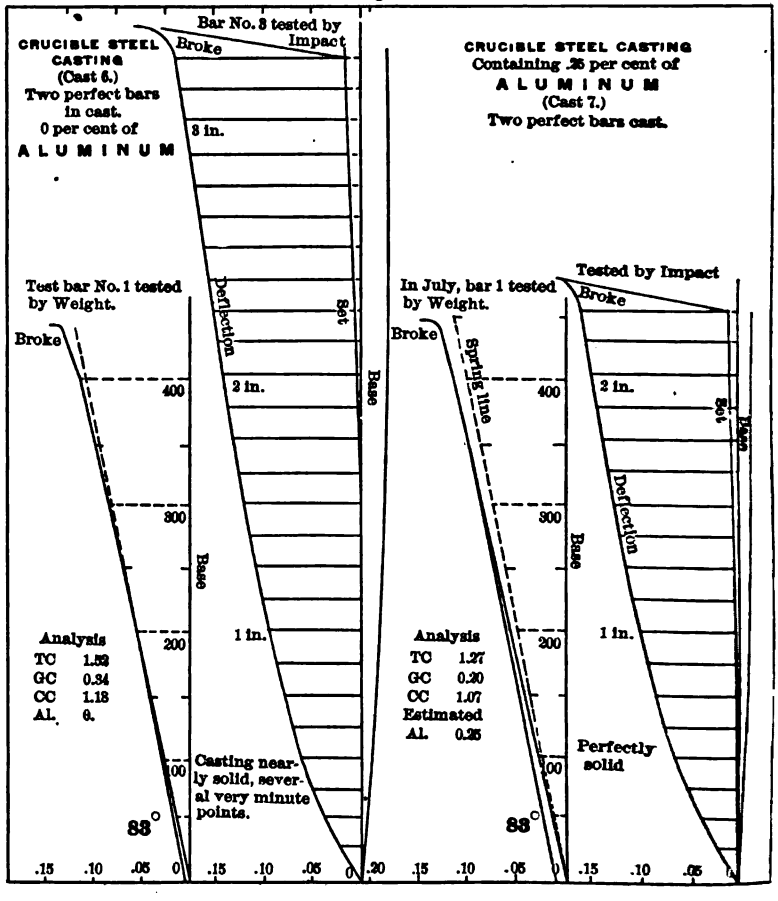
for one and 0.261 for the other; average, 0.255½. The ultimate strength is 440; proportional up to 225 pounds; deflection, 0.135; elasticity, 0.115; set, 0.02; rigidity, 83°; hardness, 40.

It sustains impact remarkably. Proportionality went up to the 1½-inch fall. Deflection was 20 and elasticity 0.175 at the breaking blow (3¼-inch fall), as against 0.135 and 0.115 for the weight-test.

Fig. 9.



Cast No. 7.—The addition of 0.25 per cent. of aluminum removes all tendency to blow-holes, but seems to make the metal less able to withstand strain. The analysis shows total carbon, 1.27; graphitic, 0.20; combined, 1.07; and it is supposed to contain about 0.25 aluminum. The diagrams are on the right of Fig. 9. The shrinkage is 0.255; ultimate strength, 450 pounds; proportionality hardly

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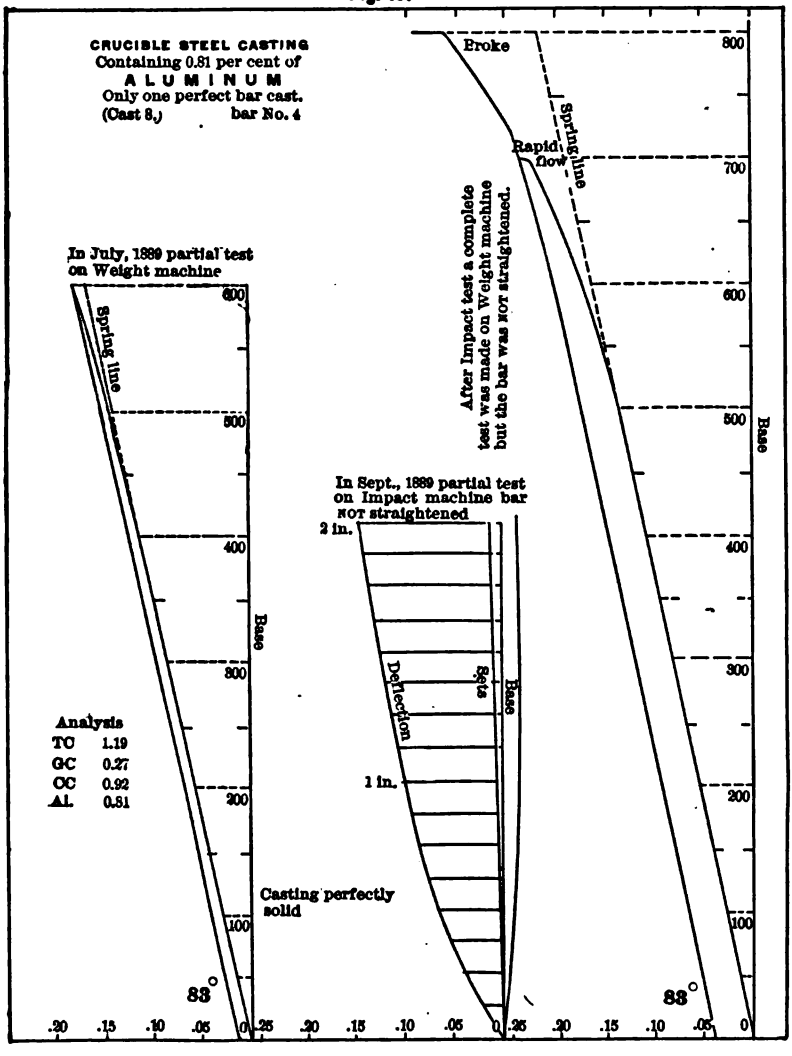
passes 50 pounds; deflection, 0.135; elasticity, 0.115; set, 0.02; rigidity, 83°; hardness, 44.

So far the deterioration is only apparent in the lack of proportionality, and this lack is also shown by the impact-test, where the

Fig. 10.



bar takes set at almost the first blow. The breaking-blow was only 2¼ against a 3¼-inch fall in cast No. 6; and the deflection is only 0.155 and elasticity only 0.145, while in cast No. 6 they were 0.20 and 0.175. The loss is, therefore, in its having taken set as soon as

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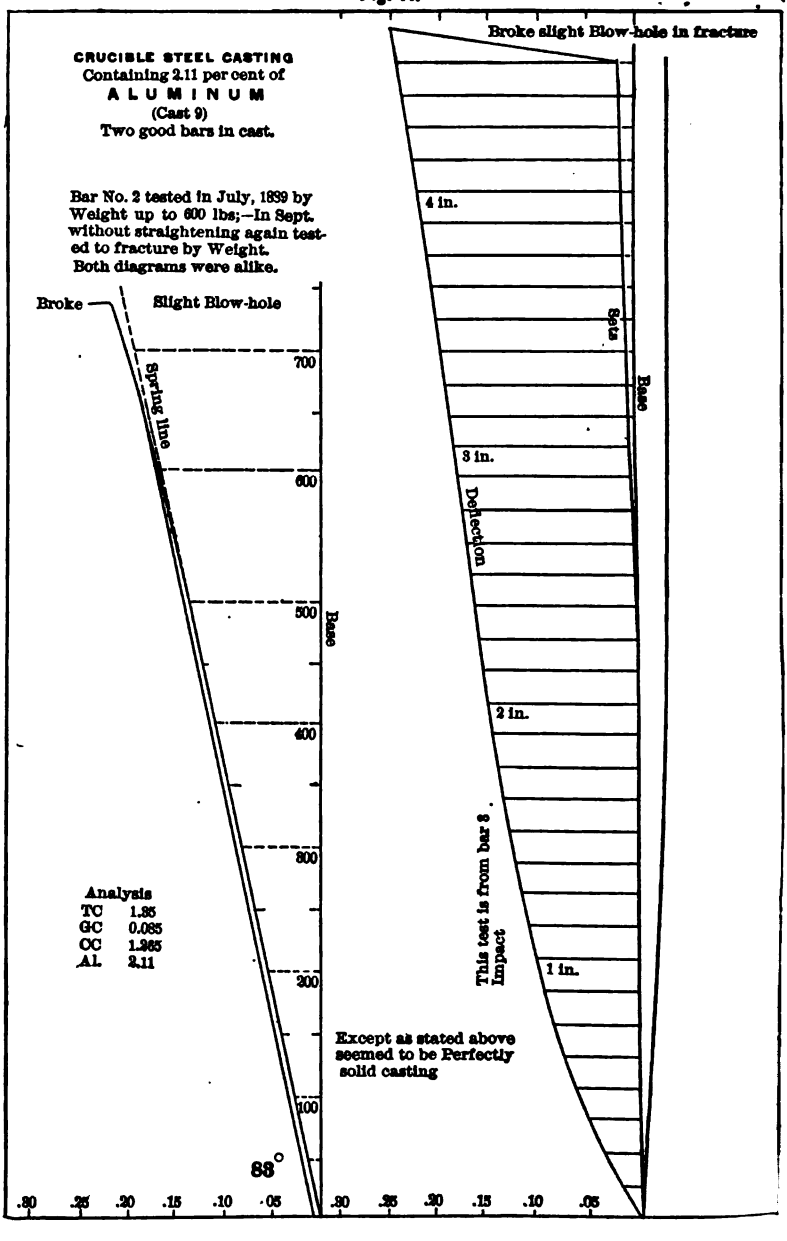
stress was applied, and in a lessened ability to withstand shock. The

Fig. 11.



texture of the steel is slightly more regular and is of finer grain.

Cast No. 8.—This cast contains Al, 0.87; total C., 1.19; graphitic,

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0.27; combined C., 0.92. We had only one bar, and first applied weight up to 600 pounds. It was proportional to 400. Rigidity was 83°. The bar rested from July 15th until September 1st, and without being straightened the same bar was tested by impact up to a 2-inch fall. It seemed to take set at the first blow. The bar was then tested without straightening by weight until rupture. Owing to the bar having been previously tested by impact, it was a spring up to the limit of previous test; therefore the proportionality to 525 is not the record for the bar. Neither is the record of deflection or set exactly correct for the metal, though it happens to be almost the same at 600 as in the first test; we may, therefore, accept these records as correct, namely: Ultimate strength, 800 pounds; proportionality, 400 pounds; deflection, 0.315; elasticity, 0.22; set, 0.095; rigidity, 83°, and hardness, 52. The shrinkage is 0.245 for the square bar. The impact-test is sufficient for comparison with casts Nos. 6 and 7. This is a remarkable metal. At 700 pounds it is quite plastic.

This diagram, Fig. 10, is a beautiful example of the stiffening which results from flow in many cases, as the spring-line extends in the last diagram 25 pounds beyond the point where the flow was arrested, and then at this point the former diagram was resumed. The grain of the metal has become quite fine.

Cast No. 9.—This metal contains Al, 2.11; total C., 1.35; graphitic, 0.085, and combined C., 1.27. We had two good bars, and the weight- and impact tests may, therefore, be compared. The shrinkage is 0.257; ultimate strength, 740 pounds; proportionality, 500 pounds; deflection, 0.322; elasticity, 0.20; set, 0.02; rigidity, 83°; and hardness, 50. (See Fig. 11.)

Under impact the bar broke at a 4½-inch blow, with deflection, 0.25; elasticity, 0.23; set, 0.02. The proportionality continued up to the 2¼-inch fall.

There was a slight blow-hole in each of the bars. The grain is white and very fine and uniform, like the very best steel.

Cast No. 10.—This contained Al, 2.52; total C., 1.39; graphite, 0.08; combined C., 1.31. The shrinkage was 0.266. We had only one bar, which did not run quite full, the corners being round. We suspected that if it had not been for the flaws in the bars of cast No. 9 the strength would have exceeded that of cast No. 8. This suspicion is proved to be correct by the test of cast No. 10. (See Fig. 12.) We first tested the bar to 400 pounds, and, finding that it had taken no set, made a partial impact-test up to a 3-inch fall, still with

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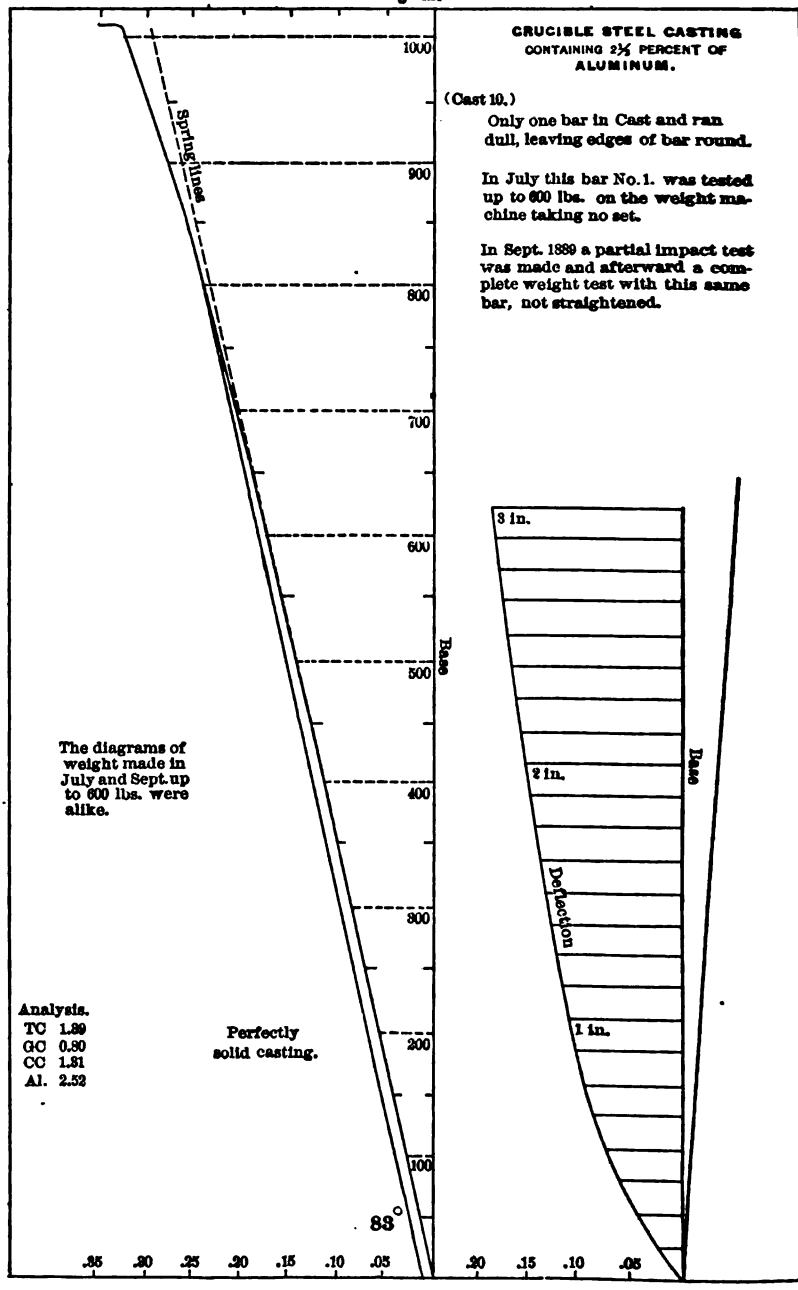
no set. We then tested it by weight until it broke. Ultimate strength, 1010 pounds; proportional to about 400 pounds; deflec-

Fig. 12.



tion, 0.33; elasticity, 0.30; set, 0.03; rigidity, 83°; hardness, 52.

This is the strongest metal in the whole series, and the surface of

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the casting was perfectly smooth, although it was run in an ordinary green-sand mould. The grain was a most beautiful white, like the very best steel.

General Remarks.—In the original bars cast after the mixture, each cast from No. 6 to No. 10 was more brittle than the one preceding it. In striking a bar a blow on the anvil, especially cast No. 10, it would fly into several pieces. Whether it was more brittle than the remelt we cannot tell, but it would seem from the records that remelting was of very great benefit, the grain being finer and the strength being increased. The records of impact are not complete enough for us to draw definite conclusions regarding the ultimate strength under shock. We think, however, that metal with small percentages of Al could resist less shock than with no Al, but that where the percentage passes 1 per cent. the ability to resist shock increases.

The influence of percentages of aluminum, less than three-quarters of one per cent., is not proved, but the indications are hardly favorable. For percentages exceeding that amount, and up to 2.50 per cent., everything is favorable. The carbon reduced the shrinkage. Aluminum has hardly any influence on shrinkage, and certainly does not lessen it. The strength is more than doubled, and increases with each addition of aluminum. Proportionality probably does not increase with the additions. Power to bend before breaking increases as aluminum increases, and this is also true of elasticity. The set is about the same for all the casts.

We give a tabulated record of results:

REMELTED WROUGHT-IRON CASTINGS.

Analytical and Physical Results of Tests of Iron Low in Carbon combined with Aluminum.

	No. of Cast.	Aluminum.	Total Carbon.	Graphitic Carbon.	Combined Carbon.	Ultimate Strength.	Limit of Proportionality.	Deflection.	Elasticity.	Permanent Set.	Rigidity.	Hardness.	Shrinkage.
Robert Ingot Iron.	Alone.....	0.00	0.35	0.04	0.31	400	250	.20	.125	.075	81°	45	.220
	With Aluminum.....	0.20	0.31	.....	.....	700	400	.15	.13	.03	84°	45	.220
Remelted Wrght Iron.	Cast 1.....	0.00	0.29	0.12	0.17	200	50	.15	.11	.04	75°	44	.292
	" 2.....	.....	0.32	0.11	0.21	250	.....	.13	.09	.04	78°	44	.281
	" 3.....	0.58	0.33	0.085	0.245	430	250	.21	.115	.095	82°	42	.273
	" 4.....	1.61	0.37	0.12	0.25	390	250	17	.10	.07	83°	41	.282
	" 5.....	2.45	0.30	0.12	0.18	590	450	.185	.116	.015	84°	30	.259

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Steel Castings.

Analytical and Physical Results of Tests of Iron High in Carbon combined with Aluminum.

	No. of Cast.	Aluminum.	Total Carbon.	Graphitic Carbon.	Combined Carbon.	Ultimate Strength.	Limit of Proportionality.	Deflection.	Elasticity.	Permanent Set.	Rigidity.	Hardness.	Shrinkage.
Blue Castings.	Cast 6.....	0.00	1.52	0.34	1.18	440	225	.135	.115	.02	83°	40	.255½
	" 7.....	0.	1.27	0.20	1.07	450	50	.135	.115	.02	83°	44	.255
	" 8.....	0.87	1.19	0.27	0.92	800	400	.315	.22	.095	83°	52	.245
	" 9.....	2.11	1.35	0.085	1.27	740 flaw.	500	.22	.20	.02	83°	50	.257
	" 10.....	2.52	1.39	0.08	1.31	1010	400	.33	.30	.03	83°	52	.266

It is remarkable that rigidity is the same, whatever the percentages of aluminum. Hardness seems to increase as the percentage of aluminum increases.

These ten casts prove that the alloy of iron and aluminum, whether carbon be present or not, is a more wonderful alloy than even aluminum bronze, and that it opens up an immense field for the use of aluminum, whenever that metal can be sold at a figure permitting the alloy to be sold at a low price.

The manner of adding the aluminum, and the fact that in each case the resultant alloy was remelted, proves conclusively that aluminum remains in the metal, and exerts the same influence on subsequent heats as it did on the first.

A statement of the elasticity, set and deflection with a weight of 200 pounds, and with impact, at a fall of 1¼ inches, for each test reported in this paper and in the papers on Aluminum in Cast Iron* and on Aluminum and Other Metals Compared†, will be found in a paper read by the author at the May, 1890, meeting of the Iron and Steel Institute (London).

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* *Trans.*, xviii., 102.

† *Trans.*, xviii., 798.

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NOTES ON THE ENERGY AND UTILIZATION OF FUEL, SOLID, LIQUID AND GASEOUS.

BY W. J. TAYLOR, PHILADELPHIA, PA.

(Washington Meeting, February, 1890.)

NATURE has furnished us with fuel in three forms, solid, liquid, and gaseous; solid, the most common; liquid, containing the greatest energy; gaseous, the most convenient for use. The tendency of the day is to the conversion of solid and liquid fuel into the gaseous form. There is magic in the word gas to many men who through ignorance or indifference imagine that by mere conversion into gas an immense quantity of energy can be added to fuel of other forms, forgetful of the law of nature, that the conversion of any substance from one form to another involves a loss of effective energy, and therefore that if in certain cases more duty can be obtained out of the gas resulting from a given amount of coal than the coal itself will supply when used direct, the cause lies solely in the more efficient utilization of the fuel in its gaseous state. Nevertheless, new processes for making gas are constantly crowded upon our notice with claims of less cost for manufacture and delivery than natural gas, and often of far more energy for the product than is contained in the coal or oil from which it is made.

I was recently asked to examine a new process of this kind. From an analysis shown me the calorific energy of the gas approximated 375 heat-units per cubic foot; on ascertaining the quantity of gas claimed to be made from a given amount of solid and liquid fuel, it appeared that for every 325 heat-units contained in the latter 900 heat-units were alleged to be delivered into the gas, or nearly *three times* as much energy as the original fuel really held.

The extravagant claim of some oil-gas advocates is still frequently heard, that by vaporizing oil with steam and then passing the mixture through a coil of hot iron pipe, an oil-water-gas containing 26,600 heat-units is formed from 1 pound of oil carrying originally 21,000 heat-units, while the only energy expended on the gas has been by the introduction of a little steam and a little extraneous heat. Theoretically, 1 pound of oil converted into water-gas carries 26,600

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heat-units, but this is only obtainable by a large expenditure of energy, the amount of which is difficult to calculate; even with apparatus of theoretically perfect efficiency, it could not be less than the quantity of heat added to the calorific energy of the oil.

Again we hear that natural gas has been entirely superseded by a wonderful process of simply superheating steam in iron pipes and getting pure hydrogen at the exit, oxygen being absorbed or carried off by some unknown and extraordinary means! And within the past year we have had reports of a marvellous grate, burning coal backward, and drawing down the chimney that carries off the products of combustion a wonderful "consuming element," which would evaporate 20 pounds of water with 1 pound of coal, thus rivaling or surpassing even a famous system of using energy over and over, by which 18 to 20 pounds of water could be evaporated to steam per pound of coal burned. These and kindred impossible schemes are constantly thrust on the investing public, often with success, and in some cases, I believe, without intention of fraud, but through the ignorance of the promoters themselves.

The cheapest artificial fuel-gas per unit of heat is common producer-gas, or "air-gas" as it might be termed, since the oxygen for burning carbon to carbon monoxide is derived mainly from air. The associated atmospheric nitrogen dilutes the carbon monoxide, making air-gas the weakest of all useful gases—that is, the lowest in combustible, both by weight and by volume. Next in the order of heat-energy comes water-gas, in which the oxygen for combining with carbon to form carbon monoxide is derived from water-vapor, and hydrogen is liberated. For equal volumes, this gas has more than double the calorific power of air-gas. Third in the ascending scale stands coal-gas, the ordinary illuminating-gas distilled from bituminous coal, which carries more than double the heat-energy of water-gas. Last, and highest in the list, comes the gas made in Nature's producer, which we cannot duplicate in practice by any known process. The calorific power of natural gas is about 50 per cent. greater than that of coal-gas. The introduction of natural gas for metallurgical purposes has largely stimulated the production and use of artificial gas made from coal and from oil, if the vapors of the latter can be fairly considered a gas.

Now, more than ever, the question of the best mode of fuel utilization is urged upon us, and it is with a view to contributing towards the elucidation of this important subject that I offer the present paper.

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In all heat-calculations I shall use the British thermal unit (B. T. U.; the heat required to raise 1 pound of water 1° Fahrenheit), and shall adopt as sufficiently accurate for present purposes the following proportions and values:

Air by weight, contains 23 parts O, 77 parts N.
Air by volume, contains 21 parts O, 79 parts N.
Air consumed in combustion:
1 pound C to CO consumes 1.33 pounds O, with 4.46 N, making 5.79 Air.
1 pound C to CO_2 consumes 2.667 pounds O, with 8.927 N, making 11.594 Air.

Heat-Units developed in burning.	For 1 pound of combustible. B. T. U.	For 1 cubic foot of combustible. B. T. U.
C to CO,	4,400	.....
C to CO_2,	14,500	.....
CO to CO_2,	4,325	319
H to H_2O,	62,000	327
CH_4 to CO_2 and water,	23,500	1007
C_2H_4 to CO_2 and water,	21,400	1593

Number of cubic feet in 1 pound of the following gases, at 62° F., and atmospheric pressure:

Air,	13.14	cubic feet per pound.
N,	13.50	" "
O,	11.38	" "
H,	189.70	" "
CO,	13.55	" "
CO_2,	8.60	" "
CH_4,	23.32	" "
C_2H_4,	13.46	" "

The heat-energies given in the above table are calculated upon the assumption of 62° F. as the initial temperature, and the reduction of the temperature of the products of combustion to the same point as the standard for the computation of all heat-energies. Of course hydrogen is usually only burned to steam, and the energy in this case at 62° initial and 212° fuel temperature, is 52,000 heat-units, or, making both temperatures 212°, about 53,000 heat-units. Many writers use this standard for hydrogen in their computations; but in all theoretical calculations we should give hydrogen credit for the energy developed when the products of combustion are reduced to the standard temperature, and compute the losses in its utilization from that standard.

The value 0.25 will be taken as a sufficiently close approxima-

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tion for the specific heat, at constant pressure, of all gases except hydrogen (for which it is 3.4). It does not appear to me necessary to burden the computations with exact theoretical figures, which add greatly to the labor without altering, for practical purposes, the nature of the results. All analyses of gases will be given by volume.

Fuel-Energetics.

Carbon-Gas.—In considering any gas-fuel, the first question is, what percentage of the energy of the fuel converted is delivered with the gas? Producer-gas, though the lowest in energy, can be produced more cheaply per unit of heat than any other. Yet in the old Siemens producer, practically, all the heat of primary combustion—that is, the burning of solid carbon to carbon monoxide—was lost, as little or no steam was used in the producer, and nearly all the sensible heat of the gas was dissipated in its passage from the producer to the furnace, which was usually placed at a considerable distance.

Modern practice has improved on this early plan, by introducing steam with the air that is blown into the producer, and by utilizing the sensible heat of the gas in the combustion-furnace. One pound of carbon, burned to 2.33 pounds of carbon monoxide, develops 4,400 heat-units, or about 30 per cent. of the total carbon-energy; in the secondary combustion, 2.33 pounds of carbon monoxide burned to 3.66 pounds of carbon dioxide develop 10,100 heat-units, or 70 per cent of the energy, making in all 14,500 heat-units for the complete combustion of the original carbon pound. Now, it is evident that if the heat of the primary combustion is not employed either to dissociate water or to impart a useful high temperature to the gas, 30 per cent. of the energy will be practically lost; *i.e.*, the gas will carry into the furnace only 70 per cent. of the total energy of the carbon. It is equally evident, that if all the heat of primary combustion could be applied to the dissociation of water, there would be no effective loss of energy in conversion; or if, instead of dissociating water, all the sensible heat of the gas (representing the heat of primary combustion) could be utilized, the loss would similarly be reduced to nil. But the complete realization of either alternative is impossible, for the loss by radiation from the producer is an important item, and the unrecovered energy expended in blowing the producer with steam amounts to from 3 to 5 per cent.

Good practice does, however, recover a considerable percentage of the heat of primary combustion by the use of both of these means,

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i.e., by utilizing the sensible heat of the gas through close attachment of producer and furnace, and by introducing with the air-blast as much steam as the producer will carry and still maintain good incandescence. In this way, about 60 per cent. of the energy of primary combustion should be theoretically recovered, for it ought to be possible to oxidize one out of every four pounds of carbon with oxygen derived from water-vapor. The thermic reactions in this operation are as follows:

	Heat-Units.
4 pounds C burned to CO (3 pounds gasified with air and 1 pound with water) develop, ...	17,600
1.5 pounds of water (which furnish 1.33 pounds of oxygen to combine with 1 pound of carbon) absorb by dissociation, ...	10,333
The gas, consisting of 9.333 pounds CO, 0.167 pound H., and 13.39 pounds N., heated 600°, absorbs, . . .	3,748
Leaving for radiation and loss, ...	3,519
	17,600

(It may be well to note here that the steam which is blown into a producer with the air, is almost all condensed into finely-divided water, before entering the fuel, and consequently is considered as water in these calculations.)

The 1.5 pounds of water liberates .167 pound of hydrogen, which is delivered to the gas, and yields in combustion the same heat that it absorbs in the producer by dissociation. According to this calculation, therefore, 60 per cent. of the heat of primary combustion is theoretically recovered by the dissociation of steam, and, even if all the sensible heat of the gas be counted, with radiation and other minor items, as loss, yet the gas must carry $4 \times 14{,}500 - (3748 + 3519) = 50{,}733$ heat-units, or 87 per cent. of the calorific energy of the carbon. This estimate shows a loss in conversion of 13 per cent., without crediting the gas with its sensible heat, or charging it with the heat required for generating the necessary steam, or taking into account the loss due to oxidizing some of the carbon to carbon dioxide. In good producer-practice the proportion of carbon dioxide in the gas represents from 4 to 7 per cent of the C burned to CO_2, but the extra heat of this combustion should be largely recovered in the dissociation of more water-vapor, and therefore does not represent as much loss as it would indicate. As a conveyer of energy, this gas has the advantage of carrying 4.46 pounds less nitrogen than would be present if the fourth pound of coal had been gasified with

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air; and, in practical working, the use of steam reduces the amount of clinkering in the producer.

A theoretical gasification of 100 pounds of carbon, deriving one-fourth of the oxygen from steam and three-fourths from air, shows the following result, in which, for the reason just given, the formation of carbon dioxide is neglected:

Process.	Products.		
	Pounds.	Cubic feet.	Composit'n by vol.
100 lbs. C burned to.....CO	233.33	3150	37.29
Oxygen derived from water, for combining with 25 lbs. carbon, liberates.....H	4.17	792	9.37
Oxygen derived from air, combining with 75 lbs. carbon, is associated with.....N	334.50	4505	53.34
	572.00	8447	100.00

Here we have 572 pounds of gas which, in our experience of the best practice, is delivered by the producer at an elevation of temperature of 600 degrees; hence, with a specific heat of 0.25, the sensible heat imparted to the gas amounts to 85,800 heat-units, or about 6 per cent. of the total energy of the carbon. If this 6 per cent. be added to the 87 per cent. first estimated, we find 93 per cent. of the carbon-energy theoretically delivered with the gas.

The quantity of heat which this gas is capable of generating by combustion is readily computed, and serves as a check on our previous calculation:

CO burned to CO_2, 233.33 pounds × 4,325 =	1,009,160	heat-units.
H " " water, 4.17 " × 62,000 =	258,540	"
Sensible heat in 572 pounds of gas, . .	85,800	"
	1,353,500	"
Solid C burned to CO_2, 100 pounds × 14,500 =	1,450,000	"
Loss in conversion, 6.65 per cent., . . =	96,500	"

The efficiency of the conversion is 93.35 per cent., showing substantial correspondence with the former result.

I have never been able to produce a gas containing as high a combined percentage of CO and H as this theoretical synthesis shows

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(46.66 per cent.), without losing temperature in the producer; and when I have spurted with extra steam and less air to make the sum of the CO and H equal to 45 per cent., the carbon monoxide ratio has always been lower and the hydrogen ratio higher, even with low carbon dioxide (say, 1 to 3 per cent.). In anthracite producer-practice, the limit of these combustibles taken together seems to be about 40 per cent., varying from 25 carbon monoxide and 15 hydrogen, to 28 carbon monoxide and 12 hydrogen—and in daily working we must at present be content to deliver only 80 to 85 per cent. of the carbon-energy, inclusive of the sensible heat of the gas.

Anthracite Gas.—Hitherto we have been treating of carbon only; considering next the gasification of anthracite coal, we find in it a volatile combustible (of which I regret never yet having seen an analysis), varying in quantity from 1.5 to over 7 per cent., and while its flame resembles that of hydrogen, the amount of marsh-gas found in anthracite-producer-gas corresponds practically with the total volatile hydrocarbons in the coal. If this is correct, all the hydrogen in the gas is derived from the dissociation of water-vapor, but this, as previously shown, is in practice higher than the theoretical quantity. We generally find 1.5 per cent. or more of marsh-gas in anthracite-gas from coal containing about 5 per cent. of volatile combustible (volatile hydrocarbons), and this proportion is about what should be expected if all the volatile combustible in the coal is marsh gas. But if it is not, where does the marsh-gas in the producer-gas come from, and how is the excess of hydrogen in the gas to be explained? If the percentage of carbon dioxide were high, and the resulting excess of heat were expended in an increased dissociation of steam, that would account for the hydrogen; but with low carbon dioxide, and all the volatile combustible represented by marsh-gas in the producer-product, it is difficult to account for all the hydrogen in the face of our assumption that we cannot gasify with steam more than one-quarter of the carbon.

If we felt confident that solid carbon and marsh-gas were the only combustibles to be considered in anthracite, it would be easy to calculate from an analysis of producer-gas the amount of energy derived from the coal, as is shown in the following theoretical gasification made with coal of assumed composition: Carbon, 85 per cent.; vol. HC., 5 per cent.; ash, 10 per cent.; 80 pounds carbon assumed to be burned to carbon monoxide; 5 pounds carbon burned to carbon dioxide; three-fourths of the necessary oxygen derived from air, and one-fourth from water.

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Process.	Products.		
	Pounds.	Cubic feet.	Anal. by vol.
80 lbs. C burned to ..... .CO	186.66	2529.24	33.4
5 lbs. C burned to .....CO_2	18.33	157.64	2.0
5 lbs. vol. HC (distilled).....	5.00	116.60	1.6
120 lbs. oxygen are required, of which 30 lbs. from H_2O liberate. .....H	3.75	712.50	9.4
90 lbs. from air are associated with.....N	301.05	4064.17	53.6
	514.79	7580.15	100.0

Energy in the above gas obtained from 100 pounds anthracite:

186.66 pounds CO, ...	807,304	heat units.
5.00 " CH_4, ...	117,500	"
3.75 " H, ...	232,500	"
	1,157,304	"
Total energy in gas per pound, . . .	2248	"
" " " 100 pounds of coal, . . .	1,349,500	"
Efficiency of the conversion, ...	86	per cent.

It will be noticed that 1.6 per cent. of marsh-gas represents all the volatile combustible in the coal, and that 86 per cent. of the total energy is delivered in the gas; but the sum of carbon monoxide and hydrogen exceeds the results obtained in practice. The sensible heat of the gas will probably solve this discrepancy, and I therefore think it safe to assume the possibility of delivering at least 82 per cent. of the energy of anthracite.

To illustrate the loss caused by forming carbon dioxide in the producer, when none of the heat of primary combustion is used for dissociating water, the following theoretical gasifications of carbon are adduced, showing the resulting gases in which 0, 5, 10, 15, 25, and 50 pounds of carbon are successively burned to carbon dioxide, and giving the percentage of energy delivered in each case, without considering the increasing proportion of nitrogen as a factor in reducing the energy-ratio of the poorer gases. (See Table, next page.)

A noteworthy relation of carbon monoxide to carbon dioxide will be observed in this table. Since one volume of the monoxide burned to the dioxide is not increased in volume, it would seem at first

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100 lbs. C, gasified with air.	Pounds of C Burned to CO_2.					
Products.	0	5	10	15	25	50
CO per cent.....	34.4	31.5	29.5	26.6	22.7	12.9
CO_2 "	.....	1.6	3.2	4.6	7.6	12.9
N "	65.6	66.9	67.3	68.8	69.7	74.2
Pounds of gas.....	679	708	737	766	824	969
Cubic feet of gas..	9183	9468	9759	10,065	10,387	12,189
Per ct. of carbon energy in gas....	70	66	63	59	52	35
Heat-units per cu. ft. of gas.....	109.7	100.5	94.1	85.8	72.04	41.1

thought that the sum of the percentages of the two should remain equal. The apparent anomaly results from the fact that nitrogen increases with the increase of carbon dioxide, reducing the aggregate percentage of the two carbon oxides.

But the formation of carbon dioxide in the producer is objectionable, not only when the heat of its combustion is lost, but even when a large portion of this heat is recovered by dissociating water. A theoretical gasification, in which 100 pounds of carbon are completely burned to carbon dioxide, and 70 per cent. of the resulting heat of combustion (1,450,000 heat units) is assumed to be recovered by dissociating water is illustrated in the following table:

Process.	Products.		
	Pounds.	Feet.	Per cent. by vol. (Approximate.)
100 lbs. C burned to.....CO_2	366.66	3153	25
70 per cent. of 1,450,000 heat-units are 1,015,000 units, which liberate from water.....H	16.34	3110	25
130.96 lbs. O, liberated from this water, combine with 49.2 lbs. C to form CO_2. This leaves 50.8 lbs. C to combine with 135.13 lbs. atmospheric O, which is associated with.....N	453	6115	50
	836	12,378	100

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Here we have only 25 per cent. of combustible hydrogen, representing 70 per cent. of the carbon energy, in 836 pounds, or 12,378 cubic feet of gas; the latter is, therefore, of poor quality, and compares very unfavorably with the 70 per cent. conversion of the all-monoxide gas in the preceding table, where 34.4 per cent. of combustible (carbon monoxide) are found in 679 pounds, or 9,183 cubic feet of gas. It follows that, whenever carbon-dioxide is formed and its heat is used for dissociating water, there is at best but a poor utilization of the energy. Probably all that can be recovered in this way does not exceed one-half of what may be obtained from carbon burned to carbon monoxide. But in special cases, where practically all the sensible heat of the gas is utilized in a non-regenerative furnace or kiln, where mechanical difficulties effectually prevent good combustion, I have found a very hot gas, containing 7 to 9 per cent. of carbon dioxide preferable to a cold gas low in carbon dioxide.

Bituminous Gas.—The first regenerative, high-temperature furnace that I ever attempted to fire with anthracite-gas started off well, but after it had reached a temperature which was a little below the required one it would heat no higher. This surprised and perplexed me greatly, for we were burning as much, if not more, coal in our anthracite-producers as had been used in soft-coal producers to heat the same furnace, while, according to our analyses, the gas carried more combustible than most bituminous gas. Under the conviction that no improvement would be effected while using anthracite only, soft coal was substituted for about one-fourth to one-third of the fuel-charge, and the furnace-temperature began to rise at once, though not as quickly as with an all-bituminous gas. In seeking an explanation for this unexpected action, it occurred to me that the appearance of the two gases in combustion was different, while their analyses were similar; and the next thought was, how can such similarity be accounted for when the coals from which the gases are made are so different?

Comparing an average analysis of Siemens producer-gas (European practice) with our anthracite-gas, and with a gas of my own analysis made from soft coal containing 55 per cent. of carbon and 33 per cent. of volatile combustible, we find: (See table, opposite page.)

The third gas in this table is made from coal containing six times as much volatile combustible as the anthracite, yet by analysis it does not carry more than twice as much hydrocarbon as the anthracite-gas. That there is a discrepancy here becomes apparent as soon

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Constituents.	Siemens Gas. (European.)	Anthracite-Gas.	Soft-coal Gas. (American.)
CO	23.7	27.0	27.0
H	8.0	12.0	12.0
CH_4	2.2	1.2	2.5
CO_2	4.1	2.5	2.0
N	62.0	57.3	56.5
	100.0	100.0	100.0

as attention is directed to it, and its cause undoubtedly lies in the fact that, whereas all the volatile combustible in anthracite is fixed gas when volatilized, only a small part of the volatile combustible in bituminous coal is fixed by volatilization, and only that fixed portion is represented in analysis. In our own case nearly all the hydrocarbon vapor was condensed in the sample cooled down to 60° for analysis, but in the passage of the gas to the furnace not enough heat was lost to condense much, if any, of it. Hence the hydrocarbon was utilized in the furnace, and effectually too, while none of it was shown in analysis. To confirm this view of the matter, it may be added that, after the two gases made from soft and hard coal are drawn or forced into a rubber bag and cooled down for analysis, no practical difference can be observed in the appearance of a jet burned from each, while great difference will be noticed if a poke-hole cover is removed from the top of either producer and the gas is lighted. Anthracite-gas burns with a sharp, blue, non-luminous flame, while that of bituminous gas is very luminous, and radiates much more heat. A theoretical synthesis will show at once, not only how much larger a proportion of the coal-energy the bituminous-gas carries, but also how much more energy by weight and volume the gas itself contains. Besides this, the heavy hydrocarbons give it luminosity in burning, which is recognized as a particular advantage by the best authorities, though one perhaps not yet generally conceded in high-temperature work.

To examine more closely into the conversion of bituminous coal, a theoretical gasification of 100 pounds of coal, containing 55 per cent. of carbon and 32 per cent. of volatile combustible (which is about the average of Pittsburgh coal), is made in the following table. It is assumed that 50 pounds of carbon are burned to carbon mon-

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oxide and 5 pounds to carbon dioxide; one-fourth of the oxygen is derived from steam and three-fourths from air; volatile combustible is taken at 20,000 heat-units to the pound—a safe assumption, I think, notwithstanding that Sir Lowthian Bell in his paper "On Gaseous Fuel," read at the Paris meeting last year, puts it at 18,000. In computing volumetric proportions, all the volatile hydrocarbons, fixed as well as condensing, are classed as marsh-gas, since it is only by some such tentative assumption that even an approximate idea of the volumetric composition can be formed. The energy, however, is calculated from weight, and is strictly correct.

Process.	Products.		
	Pounds.	Cubic feet.	Per cent. by vol.
50 lbs. C burned to.....CO	116.66	1580.7	27.8
5 lbs. C burned to.....CO_2	18.33	157.6	2.7
32 lbs. vol. HC (distilled).....	32.00	746.2	13.2
80 lbs. O are required, of which 20 lbs., derived from H_2O, liberate.....H	2.5	475.0	8.3
60 lbs. O, derived from air, are associated with N	200.70	2709.4	47.8
	370.19	5668.9	99.8

Energy in 116.66 lbs. CO.....504,554 heat-units.
" 32.00 lbs. Vol. HC.....640,000 "
" 2.50 lbs. H.....155,000 "

1,299,554 "
Energy in coal.....1,437,500 "
Per cent. of energy delivered in gas..... 90.0
Heat-units in one pound of gas 3,484

When these figures are compared with the theoretical gasification of anthracite, the vastly greater energy, both by weight and volume, in the bituminous gas is seen at once. It is worth even more in practice than appearance indicates, since the high percentage of hydrocarbons is associated with lower nitrogen. All of the 32 per cent. of volatile combustible except the tarry matter must be volatilized and utilized in its full strength, whether it be fixed gas or simply distilled hydrocarbon; for this purpose it should not be suffered to cool below 300° before it enters the combustion-chambers or regenerators; the higher its temperature at the furnace the better.

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After studying the comparative figures it is no longer surprising that more work, particularly in high temperatures, can be done with bituminous- than with anthracite-gas. But why the discrepancy in the analyses has never before been discovered or evoked comment is strange. Subsequent to my investigations I found that at all times when anthracite-gas was used in regenerative furnaces, it was necessary to gasify in the producers from two to three times more coal per ton of iron heated than when bituminous-gas was used. It is also well known that rate and effectiveness of heating rises with the percentage of volatile combustible, and that relatively less coal is used with high volatile combustible. One of the large iron companies is now, however, erecting a plant to test thoroughly the practicability of anthracite-gas in high-temperature regenerative furnaces. The results may prove that it can be used advantageously, especially when supplemented with a little oil, which could be introduced into the furnace about where the air and gas unite, and thus secure a luminous hydrocarbon flame. Such use of oil is said to be practiced to a limited extent in Europe, as a supplement to water-gas. Broadly speaking, and for a wide field of work, the quality of the heating that has been done with anthracite-gas is good. The comparison with bituminous-gas is not always as unfavorable as the one we have considered. The energy of the bituminous-gas described was 3484 heat-units per pound, as against 2246 heat-units for the anthracite; but most bituminous coals are much lower in volatile combustible and higher in carbon than our specimen coal. Possibly a fair average would be 70 per cent. of fixed carbon and 20 per cent. of hydrocarbon with 10 per cent. of ash. A theoretical gasification of 100 pounds of such a coal, burning 5 pounds of carbon to carbon dioxide, and deriving one-fourth of the oxygen from water and three-fourths from air would show this result. (See table, page 872.)

When gases are cooled to the temperature of the atmosphere before use, the data for comparison are entirely altered, and unless in that case the bituminous vapors can be fixed in the top of the producer the anthracite-gas becomes the more valuable.

Mixed Gas.—Another plan in producer-practice has been suggested as advantageous, namely, alternately blowing air and steam, and then mixing the resulting gases in a holder. A moment's reflection would seem all that is necessary to convince any one that there could be no possible advantage in this, as the heat produced and absorbed must be the same practically as in the case of running the producer continuously, carrying in the air-blast all the water vapor which the

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Process.	Products.		
	Pounds.	Cubic feet.	Per cent. by vol.
65 lbs. C burned to.....CO	151.6	2054	30.8
5 lbs. C burned to.....CO_2	18.3	157	2.3
20 lbs. Vol. HC (distilled).....	20.0	466	7.0
25 lbs. O, from water, liberate.....H	3.1	588	9.0
75 lbs. atmos. O mixed with .....N	251.2	3391	50.9
	444.2	6656	100.0

Calorific energy of the gas.....1,247,870 heat units.
" " " per pound... 2,809 "
" " " coal.....1,415,000 "
Efficiency of the conversion.....88 per cent.

producer will take while maintaining good incandescence. But it is really a disadvantage, as all sensible heat must be lost in passing the gas to a holder and mixer, entailing a further loss in condensation of hydrocarbon vapors. Besides this, there is a curious action which takes place in a bed of highly incandescent coal when all steam is put on, well known to every water-gas man. The production of water-gas is very rapid and complete for the first few minutes; but long before the bed of fuel has lost enough heat to stop the dissociation of water-vapor, the gas will be found to contain a very large percentage of steam, which will continue to increase till it is nearly all steam, while the fuel-bed is still at a good heat. It would seem from this, that there is a coating of some kind soon formed on the fuel, that prevents the oxygen of the water from combining with the carbon, and that does not form at the same temperature when both air and steam are used. Some engineers think this is due to a sort of fusion of the ash, making a thin coating on the surface of the coal, while others charge it more to a rapid cooling of the surface, or both. But, be this as it may, I am very certain that when the temperature of the fuel-bed of a producer is too low to make water-gas alone, it is plenty hot enough for making gas with air and steam together.

From this, it is evident that more water-vapor can be dissociated in a continuous than in an intermittent process. Hence a richer gas and more of the energy of fuel delivered; besides the other advantage named, the utilization of the sensible heat of the gas and of the hydrocarbon vapors.

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Water-Gas.—There is much more literature at our command on water-gas, than on producer-gas. It is made, as is well known, in an intermittent process, by blowing up the fuel-bed of the producer to a high state of incandescence (and in some cases utilizing the resulting gas, which is a lean producer-gas), then shutting off the air and forcing steam through the fuel, which dissociates the water into its elements of oxygen and hydrogen, the former combining with the carbon of the coal, and the latter being liberated.

This gas can never play a very important part in the industrial field, owing to the large loss of energy entailed in its production; yet there are places and special purposes where it is desirable, even at a great excess in cost per unit of heat over producer-gas; for instance, in small high-temperature furnaces, where much regeneration is impracticable, or where the "blow-up" gas can be used for other purposes instead of being wasted. Some steel-melting has been done in Europe with this gas, under the claim that much more work can be gotten out of a furnace in a given time, owing to the greater energy of the gas, so that the extra cost is more than balanced. I doubt this, because of the lack of luminosity (hydrocarbon flame), unless some oil is introduced into the furnace, as before described.

I will now consider the reactions and energy required in the production of 1000 feet of water-gas which is composed, theoretically, of equal volumes of carbon monoxide and hydrogen.

	Pounds.
500 cubic feet of H weigh, ...	2.635
500 cubic feet of CO weigh, ...	36.89
Total weight of 1000 cubic feet, ...	39.525

Now, as carbon monoxide is composed of 12 parts carbon to 16 of oxygen, the weight of carbon in 36.89 pounds is 15.81 pounds and of oxygen 21.08 pounds. When this oxygen is derived from water it liberates, as above, 2.635 pounds of hydrogen. The heat developed and absorbed in these reactions (roughly, as we will not take into account the energy required to elevate the coal from the temperature of the atmosphere to say 1800°) is as follows:

	Heat-units.
2.635 pounds H absorb in dissociation from water 2.635 × 62,000, ...	=163,370
15.81 pounds C burned to CO develops 15.81 × 4400, .	= 69,564
Excess of heat-absorption over heat-development, .	= 93,806

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The loss due to this absorption must be made up in some way or other.

Now, if this could be added from carbon burnt to carbon dioxide without loss by radiation, we would only have to burn an additional 4.83 pounds carbon to supply this heat, and we could then make 1000 feet of water-gas from 20.64 pounds of carbon (equal 24 pounds of 85 per cent. coal). This would be the perfection of gas-making, as the gas would contain really the same energy as the coal; but instead, we require in practice more than double this amount of coal, and do not deliver more than 50 per cent. of the energy of the fuel in the gas, because the supporting heat is obtained in an indirect way and with imperfect combustion. Besides this, it is not often that the sum of the carbon monoxide and hydrogen exceed 90 per cent., the balance being carbon dioxide and nitrogen. But water-gas should be made with much less loss of energy by burning the "blow-up" (producer) gas in brick regenerators, the stored-up heat of which can be returned to the producer by the air used in blowing-up.

I think too little attention has been given to returning lost heat by the air and too much to superheating steam; but from these two sources much lost heat can be returned to the producer, resulting in much saving of energy. I will here remark that when we make fuel-gas for domestic use, water-gas must necessarily play an important part. Hence, the importance of recovering as much as possible of the energy now lost.

Fuel-Oil.—The average yearly production of petroleum for the last ten years in this country was, say, 24,165,920 barrels, equal to 3,310,400 tons, against 150,000,000 tons of coal mined in 1889. Now, as the energy of oil is practically 50 per cent. more than that of coal, if all the oil taken from the ground last year had been used for fuel it would have displaced on this basis 4,965,600 tons coal only; but assuming that oil could deliver in practice double the energy of coal, it could then displace only 6,620,800 tons, and we would still require 143,379,200 tons for heat. So that oil cannot play an important part in supplying our heat-requirements. The natural gas used in 1889, it is estimated, contained energy equivalent to from 12,000,000 to 15,000,000 tons of coal, or more than twice the energy of the oil-production of the country for the same time.

But as before stated, oil contains so much more energy, particularly in proportion to its volume, than any other available fuel, that it is a valuable heating-agent in some special and high-temperature furnaces.

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Common crude petroleum is composed of about 84 parts carbon and 14 parts hydrogen; the balance (2 parts) being earthy matter. Hence the energy per pound is approximately:

C to CO_2 .84 × 14,500	= 12,180
H to water .14 × 62,000	= 8,680
	20,860

or 44 per cent. more than that of a pound of good coal, which, owing to the hydrocarbons in it, usually carries the energy up to what it would be if it was pure carbon, and in some cases more. Oil can be burned with less relative waste than coal, but the best evaporation with oil in practice has never exceeded coal by more than about 50 per cent. The barrel of petroleum of commerce is 42 gallons, weighing 6½ pounds per gallon.

It is the enormous amount of energy in a cubic foot of oil that makes piping it for long distances practicable. It is often asked why we cannot pipe gas for hundreds of miles as well as oil. When it is considered that 1 cubic foot of oil carries about the same energy as 1000 cubic feet of natural gas at atmospheric pressure, it will be readily understood why transporting fuel in the form of natural gas, in long-distance pipe-lines, is entirely impracticable. Without compression and without acceleration of travel, the line would have to be about one thousand times larger to transport the same energy, and for water-gas more than three thousand times larger. This can be reduced by compression and speed (which would increase the friction), but it would still be much too costly to be practicable.

Fuel-Utilization.

Boiler-Firing.—There is no heating where more of the energy is made available than in the evaporation of water in a good boiler. Fifteen pounds of water evaporated from and at 212° to steam at atmospheric pressure is the theoretical limit for 1 pound of good coal, equal to, say, 100 parts of pure carbon.

To evaporate, in direct firing, 10 pounds of water from 1 pound of coal is not unusual practice, and 11 or 12 pounds under exceptionally favorable circumstances is not entirely beyond our reach. Twelve pounds would be a utilization of 80 per cent., and 10 pounds 66⅔ per cent. of the energy in the fuel. Compare this with an iron puddling-furnace, which in old practice is estimated to utilize about 3 per cent. of the energy, and we have a fair comparison of the two

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extremes. In one case, we send the hot combustion-products to a chimney at practically the same temperature as the furnace (which is high) and in the other, we discharge them at a comparatively low temperature. That is, if the temperature of the combustion-chamber is 2000, and that of the smoke-stack 500, we have utilized just 75 per cent. of the energy of the fuel, provided we have perfect combustion without the introduction of any excess of air. This is impossible in practice as yet.

It is a great mistake to suppose that slow combustion under a boiler and a consequent low temperature is economical; for the greater the difference of temperature between the fire-box and chimney, consistent with complete combustion, the greater, of course, is the utilization of heat.

To further illustrate this, if we could increase the temperature of combustion from 2000 to 4000 without increasing the temperature of the chimney gases above 500, we would only be losing one-eighth of the energy, instead of one-quarter; and again, if the fire-box were 1000 and the chimney 500, the loss would be one-half.

The three points to be considered, then, for the utilization of fuel-energy under boilers are: first, perfect combustion; second, to get this with the least possible excess of air; and third, to maintain the greatest difference possible between the fire-box and chimney.

We will now consider from a theoretical standpoint the advantages and disadvantages of gas-firing boilers against direct-firing. I have done some experimenting in this line, but without the best arrangements of either producer or boiler; and the results were quite satisfactory in some respects. The evaporation was $10\frac{3}{10}$ pounds of water with anthracite, and $10\frac{6}{10}$ pounds with bituminous coal, from 212° to steam at atmospheric pressure. In both cases the coals were good.

But I do not think we shall do any firing of boilers with gas from anthracite, as there is so much refuse, such as culm and small buckwheat, that can be burned to advantage on a grate with a thin fire, but cannot be gasified in a producer to advantage, where a deep fuel-bed is necessary. Moreover, the volatile combustible is low, and, being all a fixed gas, can be burned. But with bituminous coal the case is different, and I think we shall eventually do considerable gas-firing with coals high in volatile combustible, for the reason that we can get more perfect combustion of the heavy hydrocarbons with very much less excess of air, thus practically doing away with smoke.

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I will add that experiments are now in hand to determine the correctness of this assertion.

In gas-firing a boiler we must debit it first with the loss by radiation from the producer and the energy required to blow it. The former can be reduced to a minimum if the walls are water-cooled and the heat of the water is utilized. For blowing the producer, 3 to 5 per cent. of the energy developed is required; but as a forced blast is much used now in direct firing, particularly with anthracite buckwheat and culm, the producer can only be charged with the difference in energy required for blowing a thick and a thin fuel-bed, and this is scarcely worth considering. Against this we have the loss due to the great reduction of temperature of the fire-box when putting on coal, and the loss due to the rush of air over the fire when the fire-door is open in both cooling and stoking, together with the loss of coal through the grate and the impossibility of burning all the hydro-carbons that are volatilized, even with a great excess of air.

It may safely be said that all these difficulties can be overcome with producer gas-firing, and the labor considerably reduced.

Between producer and mechanical stokers, some of which are approaches to producers, rests the successful solution of the problem of a better utilization of the energy of the coals high in volatile combustible, and the avoidance of the smoke nuisance.

Much has been said of late about a down-draught producer for making both water- and producer-gas from bituminous coal. I am inclined to the belief that this plan may be desirable for making water-gas from these coals, yet I doubt if it will be available for producer-gas alone. The hydrocarbons are, of course, all fixed in their passage down through the incandescent fuel; but, more than that, they are broken up at the same time, hydrogen being liberated and the carbon combining with oxygen from the air forming carbon monoxide and dioxide, and the latter seems almost invariably high. In consequence of this the gas is not only non-luminous, but the energy is low, like gas from anthracite. An excessive quantity becomes necessary in high-temperature furnaces, compared with the gas high in hydrocarbons.

The modern practice of heating by radiation instead of by contact is undoubtedly right; hence the high roof of regenerative gas furnaces, and the large volume of luminous gas with its powerful radiating properties over the bed of iron. It is certainly a fact that we require much more than double the volume of non-luminous gas

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than we do of luminous to do a given amount of heating at high temperatures.

Quite recently, a new system for a gas-furnace, designed by members of the Siemens staff, has been used to some extent in England. The plan is to divide the hot gases from the furnace, sending part to the chimney direct through the air-regenerators, and part to the producer for recarbonizing,—in other words, using the gas over and over. It might seem at first glance that this would be a saving of just one-half of the fuel, as it would only require one-half as much carbon to make carbon monoxide from carbon dioxide as from air and carbon direct; but when we consider that to reduce carbon dioxide to carbon monoxide absorbs the same amount of heat that is developed in burning carbon monoxide to carbon dioxide, it will be seen that, if there is any advantage to be gained in the plan, it can only be from a better utilization of the waste heat from the furnace, which is used to counterbalance the absorption of heat in the producer. It occurs to me also that there are some objectionable features in this plan of introducing the hot gases from the furnace into the bottom of the producer, as they would be hot enough to destroy anything in the shape of an iron conduit or grate bars. The statements made by these gentlemen as to the amount of coal used to heat one ton of iron or steel show good results, but not much, if any, better than we are getting in good practice in regenerative furnaces in many works in this country.

The advantages of regenerative furnaces have also been questioned latterly. Sir Lowthian Bell, in his paper at the Paris meeting, took the ground that the best utilization could be made by using the waste heat of the furnace for making steam. But, if there were no other objections to this, the waste heat from the furnace, heating iron for instance, would be very much more than is necessary for furnishing the power to roll the product. For this reason alone it is far better to recover the waste heat and return it to the furnace, generating steam in a separate apparatus as required; for it will be impossible to arrange any works so as to utilize all the waste heat direct from furnaces.

Regenerative furnaces have been much improved of late years by making the roofs higher and working on the radiating principle. Maximum economies can only be obtained from these furnaces, however, by running them continuously, say for a week at a time, as it takes a large expenditure of energy to heat them up when they are once allowed to cool.

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Probably the best regenerative furnace that can be built is a long one, fired at one end and charged continuously at the other, the piles or billets being rolled forward towards the hot end. A furnace built in this way long enough to reduce the temperature of the escaping gases to 600° or 800° has been found economical. One of the large Pennsylvania steel-mills has had several of these in use for some years, heating cold billets on about 200 pounds of coal per ton of steel. Against this low fuel, however, must be charged the considerable additional labor for working the billets forward.

The character of the coal used in producers for heating regenerative furnaces has much to do with the economy. The extreme variations I have found to be, the minimum 200 pounds and the maximum 1800 pounds, per ton of iron or steel heated; and it will invariably be found that the volatile combustibles were the highest when the pounds of coal consumed were the lowest. This shows the importance of giving especial attention to the character of the coal, both bituminous and anthracite, used in the producer, as well as to producer-practice. If it can be avoided, no coal should be used, the ash of which fuses at a very low temperature. The fusibility of the ash limits the maximum rate of combustion in a producer; and I have found in practice that with a coal having the most infusible ash, about sixteen pounds per hour is the maximum amount that can be gasified per square foot of area of fuel-bed. But with a fusible ash, the rate of combustion will have to be brought down to from five to seven pounds per square foot, to make good gas continuously without excessive labor or much waste. I have found that the ash of some anthracites would fuse at so low a temperature that the whole fuel-bed would soon mat together on too rapid combustion, and stop gas-making altogether.

Another important point when anthracite is used is, that too much fine dust is very objectionable, as it makes the interstices too small, or much smaller in some parts of the bed than others. This tends to "honeycomb" the fuel in combustion unless much barring is done. Or what is still worse, if the resistance in the fuel-bed is too great, the blast will seek the walls as the place of least resistance, and the gas will be worthless.

I find in practice that No. 1 buckwheat is the best size for anthracite producer-practice, although pea will do very well.

In the case of caking bituminous coals, the fine coal cokes into large masses, which, on being broken with a bar, make the fuel-bed porous and open.

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In the construction of a producer, the important point, aside from proper proportions, is to be able to work and clean it without wasting coal or losing time and without excessive labor; then by paying the men in charge in proportion to the carbon dioxide found in the gas (frequent analyses of which can be readily made), making their wages high on low carbon dioxide and *vice versa*, very good work is obtained. This has been the practice in some works, I understand, to a considerable extent, with very satisfactory results.

The importance of low carbon dioxide has been previously noted and even if we recover this energy in the dissociation of water, as a diluent it is more objectionable than nitrogen.

Kiln-Firing.—In many direct-fired kilns and furnaces, the construction is such that it is difficult, if not impossible, to secure a good utilization of the energy, owing to imperfect combustion, difficulties in firing and stoking, losses incident thereto, and so forth. This is notably the case in sugar-house char-kilns, pottery-kilns, brick-kilns, etc.; and producer-gas, properly made and applied to this kind of work, will show advantages and economies, even without much if any regeneration. I have known as much as 55 to 60 per cent. saving in fuel made in gas-firing with poor regenerators, over direct firing at the same works, showing of course that the utilization of the fuel in direct firing was very poor. But regenerators returning as much of the waste heat as possible should invariably be used when at all practicable; and when high temperatures are required they are indispensable.

Gas-Engines.—The improvements in gas-engines and their adaptation to producer-gas are rapidly developing the importance of a more direct use of fuel-energy; and the power developed with producer-gas in this way in the past few years in Europe, is a forcible illustration of what can be done in this line. One and one-quarter pounds coal per I. H. P. is now common practice; and there is little doubt that this will shortly be reduced to less than one pound by the introduction of improvements in both producer- and engine-practice.

The importance of this system of power, and its economies and advantages where a constant power is required, will be far-reaching, especially in electric lighting, pumping water, etc., as the best type of a triple-expansion engine, with its high costs and complications, coupled with the best boiler-practice, cannot go much below, say, 1½ pounds of coal per I. H. P.

For convenient reference I insert here the following table showing

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what may be considered average volumetric analyses, and the weight and energy of 1000 cubic feet, of the four types of gases used for heating and illuminating purposes.

	Natural Gas.	Coal-Gas.	Water-Gas.	Producer-Gas.	
				Anthra.	Bitu.
CO	0.50	6.0	45.0	27.0	27.0
H	2.18	46.0	45.0	12.0	12.0
CH_4	92.6	40.0	2.0	1.2	2.5
C_2H_4	0.31	4.0	.....	.....	0.4
CO_2	0.26	0.5	4.0	2.5	2.5
N	3.61	1.5	2.0	57.0	56.2
O	0.34	0.5	0.5	0.3	0.3
Vapor	.....	1.5	1.5	.....	.....
Pounds in 1000 cubic feet	45.6	32.0	45.6	65.6	65.9
H. U. in 1000 cubic feet	1,100,000	735,000	322,000	137,455	156,917

THE RIGHTS OF THE OWNER OR POSSESSOR OF A LODE MINING-CLAIM.

BY CARL HENRICH, NOBLE, ILLINOIS.

(Washington Meeting, February, 1890.)

I. The Location of Lode-Claims.

The legal requirements concerning the location of a lode-claim are mainly contained in Section 2320 of the Revised Statutes, which reads as follows:

"Mining claims upon veins or lodes of quartz or other rock in place, bearing gold, silver, cinnabar, lead, tin, copper or other valuable deposits, heretofore located, shall be governed as to length along the vein or lode by the customs, regulations and laws in force in the State of their location. A mining-claim located after the tenth day of May, 1872, whether located by one or more persons, may equal, but shall not exceed, 1500 feet in length along the vein or lode; but no location of a mining-claim shall be made until the discovery of the vein or lode within the limits of the claim located. No claim

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shall extend more than 300 feet on each side of the middle of the vein at the surface, nor shall any claim be limited by any mining regulation to less than 25 feet on each side of the middle of the vein at the surface, except where adverse rights, existing on the tenth day of May, 1872, render such limitation necessary. The end-lines of each claim shall be parallel to each other."

The discovery of a "vein or lode" within the limits of the claim, is made a pre-requisite of the location of a lode-claim. No express definition of a legal lode or vein is given in any section of the Revised Statutes. The only requirement of a lode contained in Section 2320, is its composition of rock in place bearing valuable mineral deposits. This section directs, however, the location of a lode-claim *along the middle* of the vein at the surface. This implies that a legal lode ought to have a recognizable extension, at its outcrop, *i.e.*, horizontally in length and width. Section 2322 speaks of the "depth" and the "downward course" of a lode. This implies a supposed recognizable "dip" of a legal lode. From these implications it would appear that the term "lode," in the original intent of the framers of the law of 1872, is applicable to those ore-deposits only, which in their shape approach more or less to the ideal plate-form of "regular" ore-deposits.

Irregular ore-deposits, with no recognizable "strike" and "dip," would thus belong to that class of mineral deposits mentioned in Section 2329, and subject to location as placer-claims. The language of Section 2329 is as follows:

"Claims usually called 'placers,' including all forms of deposit, except *veins** of quartz or other rock in place, shall be subject to entry and patent," etc.

If every valuable ore-deposit which is rock in place were to be regarded as a legal "vein or lode," Section 2329 ought to read:

"Claims usually called 'placers,' including all forms of deposits which are not rock in place, shall be," etc.

This use of the word "veins" in Section 2329, excepting only "veins," or lodes, from its operation, is an additional proof that a legal lode ought to have these recognizable characteristics of "strike" and "dip," implied and mentioned in the sections of the law relating to lode-claims.

Of course, as with all other natural objects, there is no sharply-drawn dividing line between regular and irregular ore-deposits. Concerning many ore-deposits there is no doubt about their classifi-

* The italics are mine.

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cation as regular or irregular. But there are others, which may with equal propriety be placed in either class.

Not only does the wording of the law indicate that the regular ore-deposits only were intended to be considered as legal lodes, but also the inherent fitness of things points in the same direction. The square location of the "placer-claim" is certainly more in accord with the location of an irregular ore-body, with all its possible ramifications and tortuous windings, impossible of determination before their actual following, than the form of a long parallelogram, which, with a length of 1500 feet, may be legally restricted to a width of 50.

The various decisions of the courts, however, have given a very liberal interpretation to the terms "vein or lode," and almost any kind of an ore-deposit, except it be contained in the "wash," slide-rock or *débris*, may be located as a "lode or vein." The term "rock in place," has been made the test of what is to be considered as a "lode or vein." Whatever may have been the original intent of the framers of the law, there is no question at present, that any form of a valuable mineral deposit, fitting the term "rock in place," may be covered with a valid lode-claim location.

After the discovery of a "lode or vein" within the limits of the claim, the section quoted *implies* that the location of the center-line of the claim shall be made coincident with "the middle of the vein at the surface." This direction for laying out a lode-claim, however, is more easily given than followed.

Section 2324, which it is useless to quote here in full, requires that any mining-claim shall be distinctly marked on the ground, and recorded, so that other intending locators in the neighborhood of the claim may be able to recognize clearly the ground already claimed. The time given to the discoverer of a lode and intending locator of a lode-claim is often too short to permit him to do the prescribed work at the place of discovery, and to ascertain, besides, the real course of the vein at the surface—the apex of the vein—for the whole 1500 feet which he wants to cover with his location. It is a rare case, that an intending locator is enabled to follow the outcrop of a lode for 1500 feet on the surface, very few veins cropping continuously for such a distance. In most cases he will find his discovered lode, or what he presumes to be such, cropping out only at intervals on the surface, and covered for the rest of the distance, more or less deeply, with *débris*. It will then happen, especially in districts where numerous intersecting or parallel fissure-veins occur (as in

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many localities of the San Juan Mountains, Colorado), that, even with frequent croppings, a locator will be deceived in tracing the apex of the lode he wishes to cover with his claim, and will locate the claim partly along one and partly along another lode. Still more difficult to follow is this requirement of the law that the claim be located along the apex of the lode, if the lode crops only at the place of discovery, or, as is sometimes the case, does not come to the surface at all, being covered everywhere by *débris*.

There are, in fact, few actual lode-claims, in which the surveyed center-line of the claim even approximately corresponds with the "middle of the vein at the surface." The location of nearly every lode-claim is defective in this respect. Some error there is, practically, always. The only question is, how great or how little the error may be.

If compliance with the requirements of Section 2320, in regard to the location of the center-line of a claim, is made difficult by the hidden or covered outcrops of normal veins, it is made entirely impossible by the shape and dimensions of the other mineral deposits which are "rock in place," and as such, have been held to be subject to lode-claim locations. Masses and ore-chambers in eruptive rocks or limestone, and many other *irregular* ore-deposits, all clearly "rock in place," may be, and have been, claimed as lodes. In such cases it is obviously impossible to locate the center-line of the claim for any number of feet *along* the middle of the vein at the surface; since the real center of the outcrop of such an ore-deposit is not a line, but a point.

The courts have wisely held that these directions of the law regarding the center-line of a lode-claim are merely advisory; that the erroneous location of a claim, so far as the coincidence of its center-line with the real course of the middle of the vein at the surface is concerned, does not invalidate the possessory right of the locator of the claim; that he will possess the claim as he located it.

The last-mentioned requirement of Section 2320 is: "The end-lines of each claim shall be parallel to each other." This provision can be complied with under any circumstances. While a culpable negligence or disregard of it is not punished by a total forfeiture of the possessory right to the claim itself, it may carry with it the forfeiture of certain possessory rights otherwise accruing to a lode-claim locator outside of his claim, and based, *at least in part*, on the parallelism of the end-lines of lode-claims.

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II. The Possessory Rights Attached to a Lode-Claim.

The possessory rights acquired by the locator, and equally enjoyed by any subsequent legal owner or possessor of a lode-claim, are mainly stated in Section 2322 of the Revised Statutes, as follows:

"The locators of all mining locations heretofore made, or which shall hereafter be made, on any mineral vein, lode or ledge, situated on the public domain, their heirs and assigns, where no adverse claim exists on the 10th day of May, 1872, so long as they comply with the laws of the United States and with State, Territorial, and local regulations not in conflict with the laws of the United States governing their possessory title, shall have the exclusive right of possession and enjoyment of all the surface included within the lines of their locations, and of all veins, lodes and ledges throughout their entire depth, the top or apex of which lies inside of such surface-lines extended downwards vertically, although such veins, lodes or ledges may so far depart from a perpendicular in their course downward as to extend outside the vertical side-lines of such surface-locations. But their right of possession to such outside parts of such veins or ledges shall be confined to such portions thereof as lie between vertical planes drawn downwards as above described, through the end-lines of their locations, so continued in their own direction that such planes will intersect such exterior parts of such veins or ledges. And nothing in this section shall authorize the locator or possessor of a vein or lode which extends in its downward course beyond the vertical lines of his claim, to enter upon the surface of a claim owned or possessed by another."

To the locator of a lode-claim, his heirs or assigns, as long as they comply with the United States, State and local requirements for holding a valid right to the located claim (*i.e.*, as long as they perform the prescribed annual labor on the claim, make affidavits to that effect and record the same, if so prescribed by State or local laws, etc.; in short, do everything necessary to keep a valid possessory title to the claim), this section grants two distinct possessory rights.

The first of these rights of possession and enjoyment is to the *surface* of the claim. There is no limitation in this section to the exclusive possession and enjoyment of the surface of the claim thus granted. In other sections of the Revised Statutes there are certain provisions as to the right of way for roads, ditches, etc., but with these we need not deal here.

The common law usually conveys, with the ownership of the surface of any piece of land, also the ownership of everything above this

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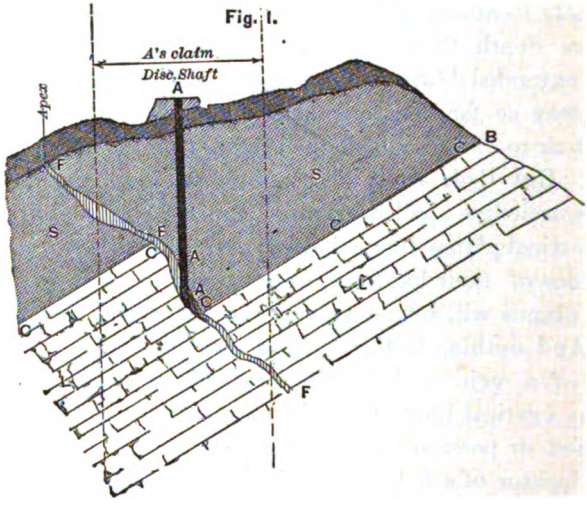
surface, and of everything below the surface, to the center of the earth. The ownership of all minerals underlying the surface of a lode-mining claim belongs by this usage to the owner or possessor of the surface, except as to that class of ore-deposits, in regard to which a different provision has been made in Section 2322. This exception to the common-law rule embraces any veins or lodes which may exist below the surface of a lode-claim. Any other ore-deposits, whether real placer-deposits or such irregular ore-deposits "in place" as might be made the basis of placer-claim locations, if they be not legal lodes or veins, contained in that portion of the earth, bounded by vertical planes laid through the boundary-lines of the lode-claim, belong to the owner or possessor of the lode-claim.

Fig. 1.



The second of the possessory rights granted to the locator of a lode-claim is the more important, since it concerns his possessory rights to *veins or lodes*, which are supposed to be the main objects of the location of such claims.

Section 2322 grants to the possessor of a lode-claim "*all* (those parts of all) veins, lodes and ledges, *throughout their entire depth*, the top or apex of which lies inside such *surface-lines* . . . ."

No distinction whatever is made between the possessory right to the vein or lode, on which the claim is supposed to be *located*, and the right to any other lode, the top or apex of which lies inside the surface-lines of the claim. Indeed, the section does not grant any possessory rights whatever *to the discovered and located lode*, except it have its apex within the surface-lines of the claim. It is

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of some importance to understand that a prospector may discover a lode within the limits of his claim, locate his claim so that the place of discovery shall be inside its boundary-lines, and still have no possessory rights whatever—at least, not by virtue of the provisions of Section 2322—to any part of the discovered lode. Fig. 1 illustrates an imaginary case of this kind.

A sinks the shaft *A A*, let us say, with the intention of prospecting the deeper portions of the contact *C C C*, between the slate formation *S*, and the limestone *L*, which contact is barren at *B*, but ore-bearing in other places. In sinking the shaft *AA*, he encounters the fissure vein *F F F*, which, at the place where he cuts it in the shaft, has a dip sufficiently steep to lead A to the belief that his claim, as laid out with the discovery-shaft in the center (according to the requirements of the surveyor-general) will include the apex of the vein. Later developments in other claims prove that no part of the apex of the vein *F F F* is within the limits of A's claim.

Section 2322 certainly does not bestow upon A any possessory rights to any part of the vein *F F*, and the stipulations of the law with regard to tunnel-rights and to lode-claims, located on discoveries made in such tunnels, do not apply to A, because he was not running a tunnel but sinking a shaft. In fact, there is no section of the United States mining law which gives to A any part whatever of the discovered lode *F F*. His lode-claim will be perfectly valid, because he has discovered a lode within its limits. He will have the exclusive right of possession and enjoyment of the surface of his claim, and also of all the parts of all veins, lodes or ledges which may have their apexes within his surface-lines; but to no part of the lode *F F*, which he aimed to secure, can A have any right of possession or enjoyment of which he cannot be deprived by the superior title of some later locator, who has covered the apex of the part to which he asserts a right. If for some reason, as will be explained later, the locator or possessor of such later claim, in which the apex of the coveted portion of the lode *F F* is situated, should have failed to acquire an extra-lateral right to that part of *F F*, then A probably would be entitled to it; for although the government has not granted him any right to it, still, in such a case, it would not have granted that right to any one else, and hence A might claim the first right of possession to it, as underlying his ground, although in reality the tenure of a lode-claim does not convey any right of possession to any lode below the surface of the claim, except as stated in Section 2322.

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Section 2322 certainly makes no distinction in the grant of possessory rights between the originally discovered and located lode and any other lode, which may have the whole or a part of its apex within the surface-lines of the claim. Any part of a vein, to be legally possessed and enjoyed by the possessor of a lode-claim, must have its apex within the boundary-lines of the claim. All which the law grants to the locator or possessor of a lode-claim, is described in Section 2322, viz., the surface of the claim, and beneath the surface all those parts of all veins which may have their apexes within his location. Nothing more. He is required to discover a lode before he can locate a claim, and he is directed how to locate his claim, if he wants to secure the largest slice of it which he is legally permitted to acquire, to wit, 1500 feet. If, through neglect, or by reason of adverse circumstances, he does not or cannot follow the directions of the law in regard to the location of his claim, as related to the discovered lode, a part of which it is supposed to be his intention to claim, the law still grants him the possession of his claim as he located it, the exclusive right of possession and enjoyment of its surface, and all those parts of lodes, the apexes of which he may have chanced to include within it. But the law does not give him the discovered lode or any parts thereof, unless he complies with the statutory directions, and includes its apex within the surface-lines of his claim.

Section 2322, in giving to the locator or possessor of a lode-claim all those parts of all veins, which have their apexes within the claim, gives him such parts of all such veins, "throughout their entire depth," although in their deeper parts they may get outside of vertical planes passed through the side-lines of the claim. But the law also enjoins him that in following, *in their downward course*, these parts of lodes which it declares he shall possess and enjoy, he must not go outside of vertical planes passed through the end-lines of the claim, and so continued in their own direction, as to intersect such exterior parts of such (parts of) lodes or ledges.

The only question which can arise in the construction of the meaning of Section 2322, is: What is meant by the words "throughout their entire depth," and "in their course downward?" In other words: What is meant by the "depth," and what, by "the downward course," of a portion of a lode?

The "entire depth" of any part of a lode is contained between two *dip*-lines or planes, passed through the ends of that part of the lode. The part of a lode, throughout its entire depth, belonging

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to a certain length or part of the apex of that lode, is obviously bounded by the two-dip lines or the two vertical planes laid or drawn at right-angles to the strike of the vein at these points, through the ends of the part of the apex in consideration. The portion of each lode bounded by such dip-lines or planes at the intersections of its apex with the surface-lines of the claim will, therefore, be the part of each lode granted by Section 2322 to the possessor of a claim, *throughout its entire depth.*

This seems clear enough, and there would hardly ever have been any doubt about this grant of extra-lateral possessory rights to these parts of lodes, had it not been for the limitation of this extra-lateral right to the space contained between vertical planes drawn through the prolonged parallel end-lines of the claim, and the peculiar phraseology in which this limitation has been expressed, to wit:

"But the right of possession to such outside parts of such veins or ledges shall be confined to such portions thereof as lie between vertical planes, drawn downward as above described, through the end-lines of their locations, so continued in their own direction that such planes will intersect such exterior parts of such veins or ledges."

Considering this provision by itself, we might be justified in assuming that Section 2322 granted to the lode-claim locator that entire part of each vein, having any part of its apex within his claim, which might be contained between the vertical end-line planes of his claim. But, as the provision is only a limitation of a possessory right already conferred, we have first to ascertain the extent of that right, *without this limitation*, and then apply the limitation to it. It is, however, obvious that if the sentence just quoted were not contained at all in Section 2322, there would never have been any doubt that only those parts of veins, having their apex within the lines of the claim, had been granted in possession, and that such parts were obviously bounded by the two dip-lines or planes at the intersection of each apex with the surface-boundaries. For they would have been granted *throughout their entire depth;* and going beyond the said dip-lines or planes would be encroaching on the deeper portion of a part of the vein having its apex *outside* of the claim, and, therefore, not conveyed to the possessor of the claim, but reserved or conveyed to the legal possessor of some adjoining claim.

In Fig. 2, we have four lode-claims, located along the two intersecting veins, $y\ y_{\text{I}}\ y_{\text{II}}\ y_{\text{III}}$ and $x\ x_{\text{I}}\ x_{\text{II}}$. No. I. locates, first, the claim $a\ b\ c\ d$, on what he supposes to be one vein; but it

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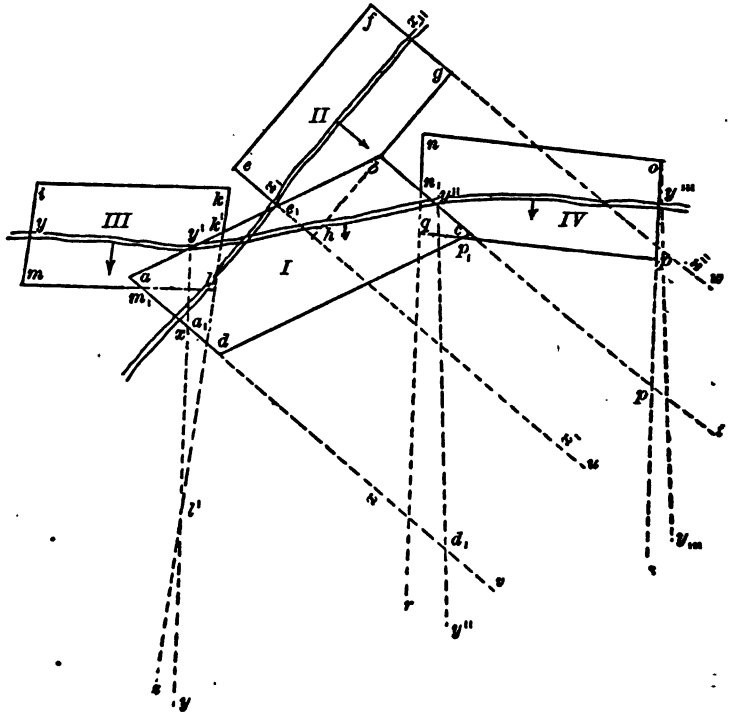
turns out to be the two veins, *X* and *Y*. Thus, his claim embraces the apex, $x\ x_1$, of the vein *X*, and $y_1\ y_{11}$, of the vein *Y*. No. II. locates the claim *e f g h*, correctly along the vein *X*, and includes the apex, $x_1\ x_{11}$, of the vein, *X*, in his claim. No. III. locates the claim *i k l m*, with approximate correctness along the vein *Y*, and includes the apex, $y\ y_1$, within the surface-lines of his claim. No. IV., finally,—almost correctly—locates the claim *n o p q*, along the

Fig. 2.



vein *Y*, and thus includes the apex, $y_{11}\ y_{111}$, of the vein *Y*, in his claim.

No. I. owns or possesses the surface, *a b c d*, and that piece of the vein, *X*, contained between the dip-planes, *x x* and $x_1\ x_1$, of that vein. In the figure, the dip-plane, *x x*, is supposed to coincide with the vertical end-line plane *a d v*; and, as the part of the vein *X*, underlying the area *v d x* $x_1\ x_1$, is inside the vertical end-line planes, *a d v* and *b c t*, no limitation of the possessory right of No. I. to the part *x x*, *throughout its entire depth*, takes place, and the extra-lateral right of No. I. to the vein *X* is confined to the part of the

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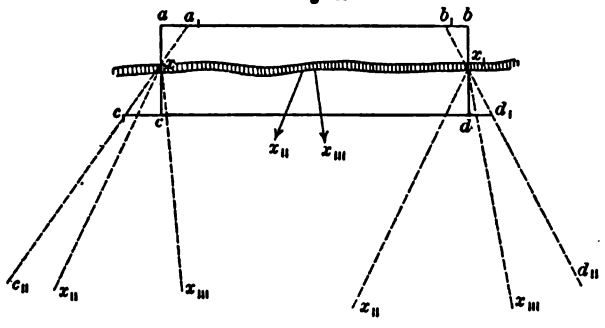
vein between the dip-line, $x_1\ x_{11}$, and the vertical end-line, $x\ d\ v$, which coincides with the other dip-line $x\ x$.

No. I. possesses, further, that part of the vein, Y, the apex of which lies inside his surface-lines, *i.e.*, the part, $y_1\ y_{111}$, throughout its entire depth, *i.e.*, as contained between the dip-planes $y_1\ y$ and $y_{11}\ y_{111}$, but limited to the space between the vertical end-line planes $a\ d\ v$ and $b\ c\ t$. No. I. possesses, therefore, that part of the vein, Y, underlying the area, $y_1\ a_1\ d\ d_1\ y_{11}$. That part of the vein, Y, underlying the area $y_{11}\ d_1\ y_{11}\ y_{111}\ p_6$, is evidently the possession of No. IV.; while that part of the vein, Y, to the left of $z\ l_1\ y_1$, is evidently the possession of No. III.

The end-line planes, $f\ g\ w$ and $e\ h\ u$, of the claim of No. II., corresponding with the dip-planes, $x_1\ x_1$ and $x_{11}\ x_{111}$, of the vein, X,

Fig. 3.



No. II. has possession of the whole length, $x_1\ x_{11}$, of the vein, X, throughout its entire depth, as contemplated by the law, as a reward for his compliance with the directions and the intent of the law in the location of lode-claims.

The parallelism of the end-lines of a lode-claim has been required evidently, with the intention to restrict the grant of the extra-lateral right of following veins in depth beyond the side-lines of a claim to a zone of equal width at all depths. This requirement of the parallelism of end-lines may have been inserted also partly to remove the temptation of making the end-lines diverge or spread towards the side of the dip of the vein, along which the claim was supposed to be located, so as to be sure to have at least one end-line coincide with, or both of them overlap, the as yet doubtful direction of the dip. Fig. 3 illustrates how diverging end-lines would benefit thus an intending locator along a vein, the dip of which was as yet unknown or doubtful.

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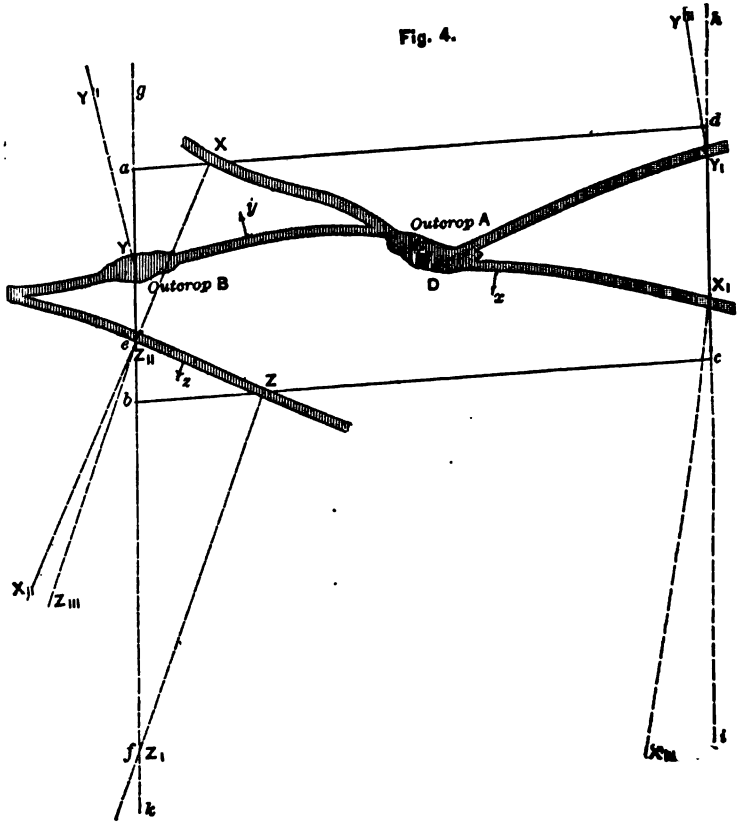
In this figure, $a\ b\ d\ c$ represents a lode-claim along the vein, $x\ x_{11}$, with as yet unknown or doubtful dip, liable to be just as well in the direction of the arrow, x_{111}, as of the arrow, x_{1111}. If there were no provision for parallel end-lines, the locator would, in this case, by locating his claim as $a_1\ b_1\ d_1\ c_1$, be sure to have the length of the vein $x\ x_1$, throughout its entire depth, whether the dip-planes at the end of the apex were in the direction $x\ x_{11}$ and $x_1\ x_{11}$, or in the direction $x\ x_{111}$ and $x_1\ x_{111}$. This would lead to the location of claims

Fig. 4.



with end-lines spreading as much as practicable in the direction of the dip of the vein sought to be covered by the claim, and would further lead to endless quarrels between neighboring claimants as to the real direction of the dip of a vein at their common apex-points. By stipulating parallel end-lines, and making these the limits of the extra-lateral right of following the veins in depth, this cause of quarrel and litigation is reduced to the least possible dimensions.

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Fig. 4 shows the applications of the provisions of Section 2322 to a lode-claim, embracing within its boundaries the apexes of several veins dipping in different directions. The claim, *a b c d*, has been located after the discovery of a lode of valuable mineral at the outcrop, *A*. The discovery shaft *D*, is sunk on the vein *X;* but in laying out the claim the outcrop *B*, of the vein *Y*, is taken for croppings of the same lode, no other prominent croppings occurring inside the claim; and while the discovery shaft is on the vein *X*, the claim really is located along the vein *Y*. The original location, as made and recorded, and under which the claim is held until patented, claims the customary 1500 feet along the discovered vein, *i.e.*, along the vein *X*. This, of course does not invalidate the possessory rights of the locator to the claim. He simply possesses the claim as he located it. The lines *a b* and *c d* are the end-lines of his claim, and the extra-lateral rights to any parts of lodes or veins having their apexes within the claim, are restricted to the space contained between the prolonged vertical end-line planes *g a b f k* and *h d c i*. Beyond these planes, Section 2322 clearly says the possessor of the claim shall not go in following those parts of veins, which he possesses by virtue of his claim, in their downward course.

These planes constitute the limitation placed on the first grant of these parts of veins *throughout their entire depth*. If these parts of veins in their downward course go outside of these vertical end-line planes, they are no longer the possession of the possessor of the claim. Section 2322 first grants to the locator of the claim *a b c d*, all those parts of lodes the apexes of which are inside of the surface-lines of his claim, *i.e.*, the part $X\ X_1$, of the vein *X*, the part $Y\ Y_1$, of the vein *Y*, and the part $Z\ Z_{11}$, of the vein *Z*, *throughout* their entire depth, *i.e.*, that portion of the vein *X*, from the dip-plane $X\ X_{11}$, to the dip-plane $X_1\ X_{111}$, these dip-planes being clearly the boundaries of that part of the vein *X*, *throughout its entire depth*, which belongs to the apex, or part of the apex, from *X* to X_1. In like manner, Section 2322 first grants to the claim *a b c d*, the part of the vein *Y* between the dip-planes $Y\ Y_{11}$ and $Y_1\ Y_{111}$, and also that part of the lode *Z* contained between the dip-planes $Z\ Z_1$ and $Z_{11}\ Z_{111}$. It further says that the possessor of the claim *a b c d* may, in the enjoyment of this grant and in following these portions of his veins, go outside the vertical planes drawn through his side-lines, *b c* and *a d*. But it then restricts him in this right of possession to the portions of these veins contained between his prolonged vertical end-line planes.

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This phraseology of the restrictions placed on the first sweeping grant of the entire depth of these parts of veins, has led to the mistaken notion that Section 2322 *granted* to the possessor of the claim *all* those portions of these veins contained between these vertical end-line planes; that these vertical end-line planes must be the boundaries *on either side* of that part of the vein belonging to the holder of the claim. They are the boundaries only in a supplementary sense. Only when the first prescribed *natural* boundaries, the dip-lines or planes at the ends of the apex, cross these vertical end-line planes; only then, and from that intersection, do these vertical end-line planes become the boundaries of that extra-lateral part of the vein belonging to a lode-claim. Thus, for the vein *X*, in Fig. 4, X_{I} X_{III}, the dip-line at the point X_{I} on that vein, forms the natural boundary on the right of that portion of the said vein belonging to the claim *a b c d;* while on the left of Fig. 4 the dip-line X X_{II} forms the boundary to the intersection-point, *e*, of the dip-line with the vertical end-line plane *g a b f k*, and at greater depth on the vein *X*, this vertical end-line plane, from the point *e* towards *f* and *k*, forms the boundary beyond which the possessor of the claim shall not go in the working of the vein, *X*.

Of the vein *Y*, in like manner, that portion underlying the area *g a Y* Y_{I} Y_{III} would belong to the claim, while its right of possession to the vein *Z* would be restricted to that part underlying the triangular piece of ground Z_{II} *Z f*.

It might be objected that the dip-line or plane of the end-point of the apex of a vein is in an unknown or uncertain direction until discovered by actual development underground, and that it would be objectionable on that account to make the dip-line or plane the boundary of the extra-lateral right of a lode-claim to the portion of each and every vein belonging to it.

To this the answer may be made that the whole United States mining law is much more uncertain than the direction of the dip of a vein, which, after all, is something fixed and unchangeable, and only unknown until developed or discovered by the opening of the vein. And it is not at all necessary to ascertain this direction before those openings in the vein are made which will permit the establishment of this boundary by survey.

But it may also be seen by a glance at the figures and diagrams given above, illustrating the provisions of Section 2322 with respect to the extra-lateral right, that, in most cases, these dip-lines or planes will be the boundaries of the extra-lateral right to a vein *only on one*

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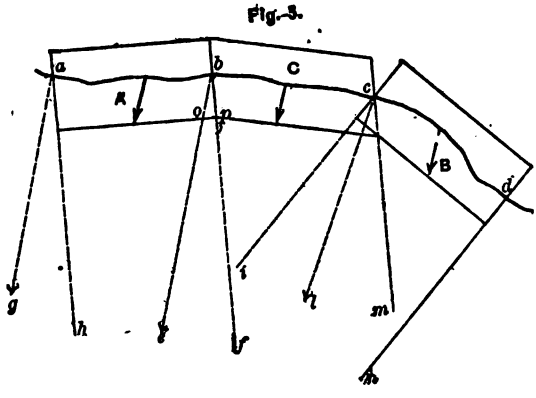
side, the boundary on the other side, in most cases, being the vertical end-line plane of the claim. This will leave, in most cases, a neutral triangular strip of ground underneath the surface, of which neither of two adjoining claims on one vein has any *legal* right of possession, but which the law positively enjoins one of the neighboring claim-owners from entering, because, in so doing, he would cross the prolonged end-line plane of his claim. Fig. 5 illustrates this more fully.

The lode *a b c d* has its apex in the three adjoining claims, *A*, *C* and *B*. A, having the portion *a b* of the apex inside the surface-lines of his claim, owns the piece *a b* of the lode throughout its entire depth, *i.e.*, between the dip-lines *a g* and *b e*, but is enjoined by Section 2322 from the enjoyment of the part *g a h* of the lode,

Fig. 5.



because lying outside of his prolonged vertical end-line plane *a h*. A has no right to the portion *e b f* of the lode, although inside the space of ground bounded by his prolonged vertical end-line planes, because the piece *e b f* of the lode has its apex inside the surface-lines of the claim *C;* and C really owns this piece, although enjoined from the enjoyment of it, because in locating the end-lines of his claim he failed to put these end-lines cross-wise to the true course, or parallel to the dip of the lode, as the law (Section 2320) advised him to do, and put them in the direction *b f* and *c m*, instead of either in the direction *b e* or *c l*, or in a direction between the two.

Had C followed the implied advice of Section 2320, and located his end-lines in the direction of the dip at either end of his part of the apex of the lode, or in the direction of the average dip of the lode, he would have secured the possession and enjoyment of his por-

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tion of the lode throughout its entire depth. As he, from ignorance or neglect, or under the force of adverse circumstances, disregarded the advice of the law in locating his end-lines, he has to suffer the consequences, and is debarred from the enjoyment of the part *e b f* of the lode. Thus the part *e b f* of the lode *a b c d* is owned neither by A nor by C.

Entirely different, however, is the question of possession and enjoyment of a part of the lode between C and B. The triangular piece of the vein lying between the prolonged end-line planes, *i c* of the claim *B*, and *m c* of the claim *C*, is divided between B and C by the dip-line *c l* of the common point *c* of their respective por-

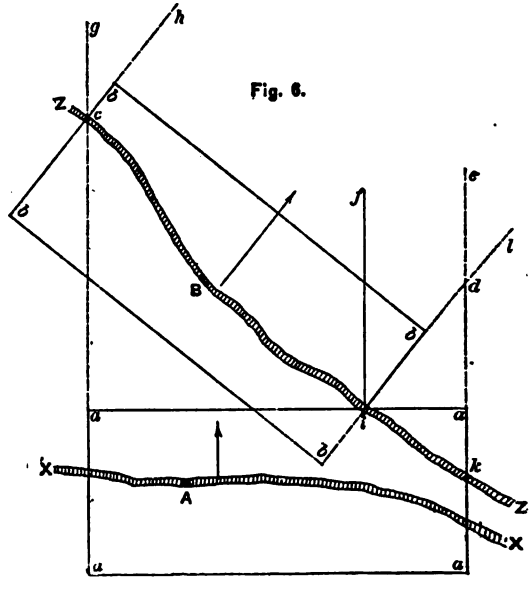

Fig. 6.

tions, *d c* and *b c*, of the apex of the lode. The direction, *c l*, of the dip-line of the point *c* of the apex cannot be determined without actual development-work on the vein; and until this is done sufficiently for ascertaining the actual direction of the common boundary-line, *c l*, it will be questionable just what parts of the triangular piece of the vein, *i c m*, belong to B and C. The dip-line of the point *c*, however, being once acknowledged as the boundary of the parts of the vein *a b c d* belonging to B and C, a compromise between B and C will be easily established, or the direction of *c l* can be easily determined by a winze or shaft sunk by both parties along the dip of the vein or lode. This cause of quarrels and law-suits will be effectually removed or diminished by the once-established recognition of the

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dip-lines as boundaries of parts of the veins belonging to a claim *throughout their entire depth.*

The equitable working of the provisions of Section 2322 of the United States mining law, when construed on the basis of the true dip of a vein as its "downward course," is best shown by its application to some hypothetical case, in which the outcrops of several veins are included in the same claim. Fig. 6 shows such a case.

A discovers the lode, *X X*, at *A*, and locates the claim, *a a a a*, along the apex of the lode, laying his end-lines parallel to the true dip of his lode, so that, to simplify the illustrations, we will assume the true dip-lines, *a e* and *a g*, at the ends of the apex of the vein, *X X*, contained in A's claim, to be the same as the prolonged end-line planes, *a e* and *a g*, of the claim, *a a a a*. B subsequently discovers the vein *Z Z* at *B*, and locates his claim *b b b b*, along the apex of the vein *Z Z*, and also puts his end-lines parallel to the dip of the vein *Z Z*. B's vein turns out to be much the richer of the two, which stimulates A to find its apex on his claim, which he succeeds in doing from *i* to *k*. The question arises now, Which part of the vein, *Z Z*, in the vicinity of the point *i* of the apex, where an especially rich ore-chute crops out, belongs to A, and which part to B? According to the view taken in this paper, B owns all that part of the vein *Z Z* enclosed between the dip-lines *i b d l* and *c b h*, while A owns only the part underlying the triangular piece of ground formed by the apex *i k*, the dip-line *i b d*, and the vertical end-line plane *k a d*. This would leave, in any case, the direction of the dip-line, *i b d l*, as the only uncertainty, and this would be a question of actual survey or of an easy and friendly compromise.

If, however, the true dip of a vein be not recognized as the "downward course" of the vein in law, then A, as the prior locator, might claim all that part of the vein *Z Z* underlying the space between the prolonged end-lines *k a d e* and a parallel vertical plane *i f*, through the point *i*, since, by following the vein *Z Z*, from the apex *i k*, in his ground between these planes, A would still be going downward.

This theory of the possessory right of A to the portion of the lode Z contained between the end line *a a e* and a vertical plane *i f*, parallel to it, has been accepted by many as the true interpretation of the meaning of Section 2322. But where in this section or in any other is found the authority for projecting this parallel vertical plane *i f*? The law only mentions the existing end-lines of the claim, which are also surface-lines of the claim, as the ones through

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which these limiting vertical planes ought to be laid. It speaks of these prolonged vertical end-line planes as the boundary lines to *all* veins, not of one pair of parallel planes for one vein and another pair for another vein. If the interpretation of the law which makes the prolonged vertical end-line planes of a lode-claim the *sole* boundaries of the extra-lateral rights of the claim to all lodes be the correct one, then A owns not only that part of the lode Z between *a e* and *i f*, but all that portion of the lode Z contained between the prolonged end-line planes *a e* and *a g*, leaving to B only that part of Z which may be above a horizontal plane drawn through the point *i* of the apex, and that part between *d e* and *d l*. This construction of the law would in this case really take from B all of his lode, although he located it correctly, simply because the prior location of A on another lode happened to cover B's location between its prolonged end-lines. Such a construction is absurd on its face. It is to avoid the absurdity of this construction, so clearly shown in this case, that recourse has been had to the projection of parallel vertical planes, as *i f* in Fig. 6, an expedient nowhere authorized in any of the sections of the mining law. Indeed, there is no interpretation of the law, except the strictly literal one, advocated herein, which will not, if strictly carried through in practice, give rise to endless confusion and flagrant unfairness and injustice.

III. Judicial Decisions and Opinions.

One of the best known decisions of the United States Supreme Court affecting lode-claims, is the "Flagstaff" decision.* In this case it was conceded that the apex of the Flagstaff lode crossed the parallel side-lines of the Flagstaff claim, or what had been surveyed as the side-lines of that claim. The Flagstaff was located under the law of 1866, which says, regarding lode-claims:

> "No location hereafter made shall exceed 200 feet in length along the vein for each locator, with an additional claim for discovery to the discoverer of the lode, with the right to follow such vein *to any depth*, with all its dips, variations and angles, together with a reasonable quantity of surface for the convenient working of the same as fixed by the local rules, etc. . . . ."

A location made under this older law was clearly a location *of the lode itself;* and it was the discovered lode which was granted irrespective to its relation to the surface, to which only an easement "for

* Cited in the paper of R. W. Raymond, "The Law of the Apex," *Trans.*, xii., 433.

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the convenient working of the lode" was given. Thus the surface-ground of the claim had really only in so far any importance under the old law, as it indicated and marked the extent of the discovered and claimed lode. And, naturally, those lines of the claim which marked, by their intersection with the course of the lode on the surface, the length of that part of the lode located and claimed by the claim as staked out and recorded, became of necessity the real end-lines of the claim. Under the law of 1866 the locator had the right of possession and enjoyment to the located part of the lode, throughout its entire depth or "to any depth" to which he chose to follow it, irrespective of the direction of his end-lines, or whether they were parallel or not.

The law of 1872, the present law, afterwards conferred on these old locations of lode-claims, where no adverse rights existed at the date of its passage, additional rights. The locators who before had only an easement to the surface of their claims, now got full and *exclusive* possession and enjoyment of the surface, and with this also the additional right of possession of all those parts of all such veins or lodes which might happen to have parts of their apexes inside the surface-lines of their claims. And the new law conferred the right to these parts of lodes throughout their entire depth even outside of the side-lines, but not outside of the prolonged parallel end-lines of the claims. It was through this additional grant of possessory rights that it became important to know, with regard to these old claims, which were the real end-lines and which the real side-lines. The Flagstaff decision settled this point for these old lode-locations. Their real end-lines are those which run *across* the course of the originally located vein or lode at the surface. If these real end-lines are not parallel, of course the grant of extra-lateral rights to veins or lodes based, in part at least, on such parallelism of the end lines, becomes void. As no law, however, is retroactive in its provisions, the right to follow the originally-located part of a lode "*to any depth*," even outside the side-or end-lines, can hardly be affected by any provisions of the later law of 1872, not in force at the time of the location of the claim. The law of 1872 also especially excepts any existing adverse rights from its operations. The boundary-planes of the parts of the originally-located lode in a lode-claim located between 1866 and 1872 are therefor clearly the dip-planes or lines of the ends of the apex of the lode, where this apex leaves the claim in question, without any restriction by the vertical end-line planes of the claim. For all other parts of all other lodes, however, except the originally-

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located lode, the extra-lateral right to such outside parts is restricted to that space of ground contained between the parallel vertical end-line planes.

To my mind, the most important passage in the decision of the United States Supreme Court in the Flagstaff case is contained in the following extract:

"If he" (the locator of a claim) "does locate his claim in that way" (improperly; in this case crosswise of the lode) "*his rights must be subordinated* to the rights of those who have properly located on the lode. *Their right to follow the dip outside of their side-lines* cannot be interfered with by him." . . . .

In this there is certainly contained a strong affirmation of the right of any locator, who has made a *correct* location such as the law contemplates, *i.e.*, parallel to the course of the vein on the surface, and with the end-lines parallel to each other and to the dip of the vein. He cannot be interfered with by any other locator, not even by the original discoverer and locator of the lode, in his right *to follow the dip* of his part of the vein. It is a pity that the testimony in this and other mining cases has never yet called to the serious attention and consideration of the United States Supreme Court the fact that *following the dip of the vein* from different parts of the apex of one vein can never bring two neighboring claim-owners into conflict. *Natural* boundaries are always best, and dip-lines or planes are such natural boundaries, capable of ascertainment by following the vein at any point of its apex (or anywhere else) in the steepest possible direction downward (or upward).

The most important decision of the United States Supreme Court regarding the established rights of claims located subsequent to 1872, is that of the "Stone-Gilt Edge" case, or the "Elgin" case, as it has been called from the name of the owners of the successful claim, the Gilt Edge. The following are a few quotations from the decision of the majority of the court:

"The surface side-lines extended downward vertically determine the extent of the claim, except when in its descent the vein passes outside of them."

This passage contains an acknowledgment of the right of the claim to everything below its surface not owned by somebody else. I must acknowledge that I cannot find any warrant for this in Section 2320 or 2322 or any other section of the statutes relating to lode-claims. But it certainly is desirable to fix the ownership somewhere, and with no adverse rights to a portion of a lode resting in

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anybody else, it would seem to belong most naturally to the owner of the surface above it. Only a Congressional enactment or amendment to the present law, giving the possessor of a lode-claim all parts of lodes not owned or possessed by another and underlying his claim, would be a *desideratum*.

"And the outside portions (of the lode) are to lie between vertical planes drawn downward through the end-lines. This means *the end-lines of the surface-locations*, for all locations are measured on the surface."

This surely tends to confirm the unchangeable character of the end-lines and side-lines of a claim as once surveyed and recorded on the ground.

"Even then, with all care possible, the end-lines marked on the surface will often vary greatly from a right angle to the true course of the vein."

This implies that a *correct* location, conferring the greatest benefit on the locator, should have its end-lines in the direction of the *dip* of the lode. The majority opinion then argues for the stability of the once-established end-lines as opposed to the minority opinion (of Chief Justice Waite and Justice Bradley), who would change the direction of the end-lines to the true dip of the vein. In combating this readjustment of end-lines, the majority of the court (through Justice Field) very justly says:

"Such readjustment at every discovery of a change in the course of a vein would create great uncertainty in titles to mining claims."

It is a pity that it was not called to the attention of the court in this case that the real boundaries of the extra-lateral right, as laid down by Section 2322, are neither rectified end-lines, drawn parallel to the dip of the vein, nor the end-line planes *by themselves*. It is a pity that the court could not have considered the proposition that the real solution of the vexed question lay between the majority and minority opinions; that while the once-established end-lines ought to be maintained in their character as well as their direction, they are really only a limitation placed on the extra-lateral grant, bounded in the first place by the planes proposed to be drawn as end-line planes by the minority of the court.

Thus I may claim, what surely not often happens, to have in this case the endorsement of both the majority and the minority opinion

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of the United States Supreme Court, to the solution of the question advocated in this paper.

One curious passage in this decision is the declaration:

"The exterior lines of the Stone claim form a curved figure somewhat in the shape of a horseshoe, and its end-lines are not and cannot be made parallel."

This declaration was hardly meant by the court to be advanced as a mathematical fact. The real meaning of the court is probably contained in the next passage:

"What are marked on the plat as end-lines are not such. The one (marked South end-line on the plat) is a side-line."

From this I take it to be the meaning of the conrt that an end-line, to be legally considered as such, must, in its extension, not cross the claim, *i.e.*, that the two vertical end-line planes must, in the requirements of the statute, include the whole extent of the claim between them; and that consequently no two lines which could be drawn at the ends of the surveyed center-line of the Stone claim could fulfil this condition and still be parallel to each other. Only in this light can be understood the curious declaration of the court, that the end-lines of the Stone claim *cannot* be drawn parallel.

Fig. 7 represents the location of the Stone claim and that portion of the Gilt Edge claim which the owners of the Stone location claimed a right to enter into in following on the dip the alleged Stone vein, with its apex as marked on the diagram.

The majority of the Supreme Court ruled that what is marked on the map as the south end-line, *v w*, is no real end-line and that through the southern end of the surveyed center-line of the claim, no real end-line can be drawn *parallel* to the north end-line; and as the parallelism of the end-lines is material to the existence of any extra-lateral right, the Stone claim people cannot follow the vein on its dip beyond the side-lines of the claim. And even if the line, *v w*, could be regarded as an end-line, the Gilt Edge claim does not fall within the space between the two prolonged end-lines.

This last remark in the majority decision would incidentally be opposed to the projection of vertical planes parallel to the end-lines at the crossing of the apex with the boundary-lines of the claim. The minority decision proposed to establish the lines *d z*, *f o*, *g p*, and *k k*, the dip-lines at the ends of the apex, as the new end-lines and boundary-lines of the extra-lateral right of the Stone claim.

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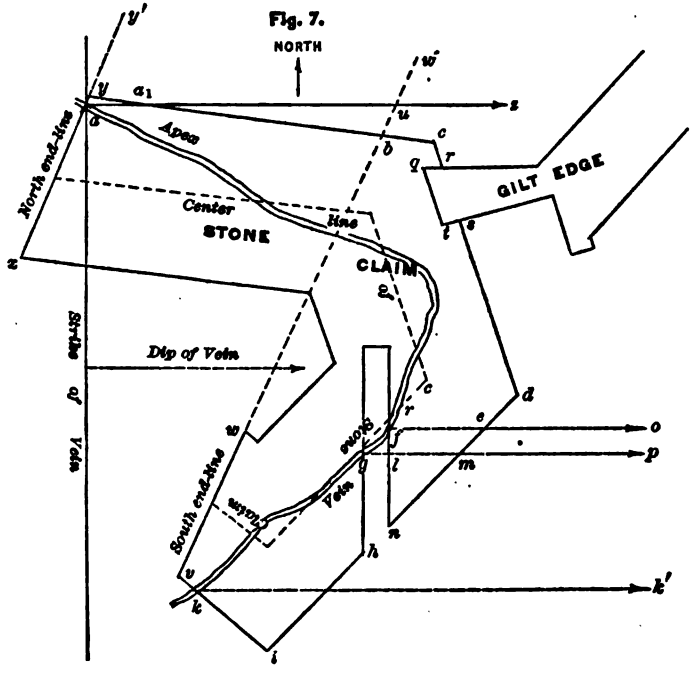
From this difference of opinion in the court, it is more than probable that if there had been two parallel real end-lines to the Stone claim, which the court could have recognized as such, the decision would have been a compromise between the two opinions, recognizing the right of the Stone claim to follow the vein from its apex downwards along the true dip of the vein, but restricting this right to such portion of the so-conveyed part of the lode as lay between the prolonged vertical end-line planes of the surveyed claim; in this

Fig. 7.



case, if $v\,w$ could be regarded as a real end-line, the part of the vein outside of the claim underlying the triangular area, $a_1\,b\,u$.

The construction of the law advocated in this paper is evidently the one adopted by Judge De Wolfe of the United States District Court of Montana in the "Silversmith"* case. Fig. 8 illustrates the points in this case. Judge De Wolfe says:

"The statute, it will be perceived, evidently contemplates that locations upon mineral veins or ledges will be made along and in the direction of the strike or course of the vein, and only provides for following the vein in its dip or downward

* Quoted in Dr. Raymond's paper, "End-lines and Side-lines in the United States Mining Law," *Trans.*, xvii., 787, *et seq.*

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course, where it passes underneath the side-lines of the claim located. It does not mention veins or ledges which cross the surface ground at right or any lesser angle, and defines in express terms the rights of the locator on veins thus crossing the surface-location and in their course or strike departing through their side-lines; but, as seen, it does provide that the locator shall have 'the exclusive right of possession of all such veins or lodes throughout their entire depth, the top or apex of which lies inside the surface-lines of the location extended downward vertically.' Now the apex of the vein in question does lie within the surface-lines of the Amy claim, ...

"The vein, by the concurrent testimony of all the witnesses, has a pitch to the north in its dip or downward course.

"What, then, is the portion of this vein which defendants have the right to follow in the dip, it being conceded in argument, and the law itself being plain on the point, that the defendant has no right to follow the vein on its strike or course after it departs or crosses the side-lines of the location on the surface? As the law gives the locator the apex within the surface-location, and the right to follow the vein on its dip or incline into the earth, is the locator not entitled to the same length along the vein on its dip that he has to the apex on the surface?"

A categorical affirmative answer to this question might be misleading. The better way, perhaps, to express in a general form, what Judge DeWolfe evidently desired to say, would have been: Is the locator not entitled to the same part of the vein on its dip, the top or apex of which part he has on the surface? If veins or lodes had always the ideal shape of a plate, and if it were not for the limitation of the extra-lateral right by the vertical end-line-planes, locators would have the right to the same length along the vein throughout the entire dip, as they have to the apex on the surface; but the cases where these conditions are fulfilled in practice are rare.

"And as the dip of a vein is supposed to be at right angles with its strike, lines drawn through the vein at right angles with its strike or course where the vein departs from the surface side-lines of the claim, and these lines extended laterally in the direction of the dip or pitch of the vein beneath the surface, and vertical lines drawn downward through these lines and intersecting the vein beneath the surface, define, in my view, the dip of the vein, and the portion thereof to which the defendant in the present case is entitled.

"This right to follow the vein in its dip is, of course, limited by the provision of the section referred to as not permitting a locator to pass beyond planes drawn vertically downward through the end-lines of the location and thence extended laterally so as to intersect the exterior parts of the vein or lode.

"It is further believed that neither of the cases referred to by the counsel and decided by the Supreme Court of the United States, wherein it is strenuously urged that a contrary doctrine has been held, asserts any principle or construction of the mining law contrary to that herein maintained; but time is not at present granted to examine or reconcile, or attempt to reconcile, those decisions with the views here expressed."

As already indicated, the only exception which I have to take to

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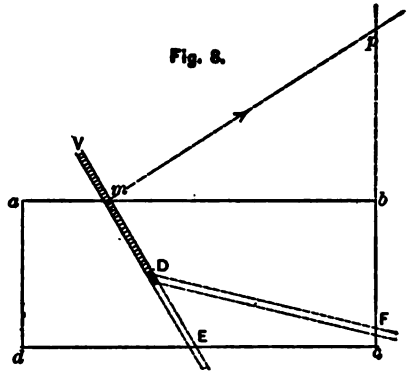
the above-quoted decision, and this an immaterial one, is contained in the passage already above commented on. The judge's question would have to be answered in the affirmative, if all veins had the ideal form of a plate. In nature, however, a vein is bounded, not by planes, but by irregular and curved surfaces at its foot- and hanging-walls; and the part of a vein included between the two dip-lines drawn from the ends of the apex of a part of a vein will vary in length at different levels. This variation may frequently be only slight, and the part of a vein belonging to a certain part of the outcrop or apex may be practically of the same length throughout its entire depth; but this is not necessarily the case. The more irregularly-shaped a vein or lode, the more its form departs from the ideal form of a plate, the more liable to variations in length will be a given

Fig. 8.



part of that vein, belonging to a certain part of its outcrop or apex, followed in its downward course or dip.

It might be argued that, even upon the conclusion that the law requires the true dip of a vein to be followed as the downward course, it would be left open to controversy, whether the local dip at the two end-lines, or the average dip throughout the claim, or the average dip of the whole lode so far as it can be traced, must be followed. It may be replied, that the *true* dip, whether we call it local or not, the *actual* dip or downward course of the vein, *as it is or exists*, at any point of the apex of that part of the vein owned by the possessor of the claim must be followed. The really-existing, not an imaginary theoretical dip must be followed. All the dip-lines,—of the true and actually existing dip of the vein,—of all those points of the apex forming together in their succession the line of apex contained within the claim, are contained, as already stated, in that part of the

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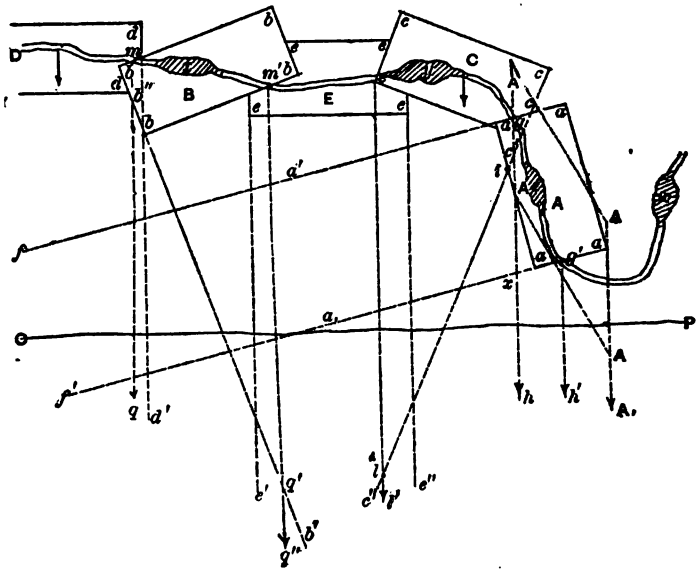
vein included between dip-lines drawn at these points, at the ends of the apex, where the apex crosses the surface-lines of the claim. As no two dip-lines of a vein can ever cross or intersect each other, the dip-lines at the ends of the apex contained in the claim are evidently the boundaries of the right of the owner of that claim to that part of the vein along its downward course or dip, the top or apex of which part lies inside of the surface-lines of the claim.

In Fig. 8, considering the line (in reality it would be probably a crooked line), *m p*, to represent the dip-line of the end-point, *m*, of the apex of the vein *V*, and *b p* to be the prolonged vertical end-

Fig. 9.



line-plane, that part of the vein *V* underlying the triangular area *m b p* would represent the extra-lateral possession to the vein, *V*, of the owner of the claim *a b c d*. This is under the supposition that the apex of the vein, in its unknown course, extending beyond *D*, would cross the boundary-lines of the claim either at *E* or at *F*, and that the dip either at *E* or at *F* would be practically the same as at *m*. It is assumed also that the claim was located subsequent to May 10, 1872, as after that date the surface-location determines the question as to which are to be considered as the real end-lines of the claim.

In further illustration of the working of the above construction

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of the law, I will take the hypothetical case, originally illustrated by Justice Beatty, and somewhat modified by Dr. Raymond, in his paper, "The Law of the Apex" (*Trans.*, xii., 436), and, finally, slightly changed by myself. I adopt the supposed state of facts given in that paper.

Fig. 9 represents the apex of a lode *A C E B D*, dipping, in a general way, as shown by the arrows in the locations, *C* and *D*, and dipping at the ends of the different parts of the apex, inclosed with the different claims, as shown by the arrowed lines of dip at these ends, to wit: the lines *m q*, m_1 *q'*, *c l*, *g k*. At *x*, *y* and *z* there are croppings upon the surface. The slightly curved line *O P* is a horizontal line drawn in the center of the lode or vein at that level, and is, therefore, the true strike line of the vein at that level, which all the dip-lines at that level must cross at right angles. The variations in the course of the apex are due partly to the fact that the strike-line of the vein is, as usual, not a straight line, and partly and mainly to the topography of the surface. The most violent of these variations is caused by a precipitous ravine, cutting through the vein, exposing the outcrop, *z*, in the face of the precipice, while on the other side of the ravine, the outcrop, *w*, shows the same vein continuing with the same general strike and dip. Between the outcrops, *x*, *y* and *z*, the course of the apex has been determined by shafts and surface-workings.

Upon this lode, A, B, C, D and E have made locations in the succession as indicated by the letters, A first and E last, placing the surface-lines of their respective locations as indicated by the boundary-lines of the claims in the diagram. What are the respective possessory rights of these locators on the vein? A, although the first comer, locates his claim, *a a a a*, in such a manner, that the downward course or the dip of his part of the vein (the part having the outcrop or apex, *g g'*, which is the part he has the right to possess and enjoy) is wholly contained in the narrow strip of ground between the dip-lines *g h* and *g' h'*. But, by locating his side-lines in the manner given in the diagram, he still further limits the part of the vein allowed him by the law, and restricts his right to the lode *A C E B D* to the small portion underlying the triangular piece of ground *g x g'*, as the law prohibits him from following the dip of his part of the vein beyond the intersection of his vertical end-line plane *a a x a'*, with the part of the vein, the top or apex of which is inside the surface-lines of his claim. His "right of possession to such outside parts of such veins .... shall be confined to such portions thereof as lie be

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tween vertical planes drawn downwards [vertically] through the end-lines of [his] location, so continued in their own direction, that such planes will intersect such exterior parts of such veins or ledges,"—is the language of the law. The law does not say that all that part of the vein shall be his which lies between his prolonged end-line-planes. It only limits the parts of veins owned by him to that space of ground. *Outside* of that piece of ground he cannot own any part of any vein by virtue of his possession of the claim. How much of any vein he owns *inside* of this space of ground, depends on the position and the length of the apex, and on the direction of the dip of any vein of which he may own a part in this ground.

In locating along the apex of the vein, from g to g', A could not have acquired any larger part of the vein from the apex $g\ g'$ downward, or that part between the dip-lines $g\ h$ and $g'\ h'$. But, by locating his end-lines parallel to the true dip of the vein, of which even very slight developments on his vein near the outcrop z, combined with the visible outcrops w, y and x, must or could have given him an indication, he might have virtually secured that part of the vein corresponding to that part of the apex of the vein located by him, throughout its entire depth. He might have done this by laying his end-lines in the direction of the dip of the lode. If he was allowed, as has been frequently claimed, to follow the lode "downward" (?) in a direction parallel to his end-lines, he would be virtually following a large portion of the vein "along its strike or course" across his side-line: *e.g.*, in working that part of the vein underlying the area $g\ a\ f\ f'$, a_1, $x\ a\ g'$, he would be following the course or the strike of the vein from his apex $g\ g'$, across his side-line $a\ a$. There is only one downward course to follow from any point of the apex of a vein, and that is by following the dip-line of the vein from that point of the apex.

The law does not propose to rectify errors of judgment made by any one in the location of claims. If A had any knowledge of veins or lodes, he must have seen from the inclination of the apex between g and g', with the necessary aid possibly of a small amount of development, that he was locating his claim practically along the dip of the vein instead of along its course; and if he, knowing or not knowing this, chose to make the location, he has to abide by the consequences of his own choice. As he was the first comer, there was no obstacle even (if, for some reason, he wanted to include the outcrop, z, in his claim) to his locating more than one claim. If, for any reason, he fails to do so, and sticks to his location, $a\ a\ a\ a$, he has to abide by

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the consequences of his own choice of locating that part of the apex. If that part of the apex, although 1500 feet long, is only in reality the outcrop of a part of the vein, 200 feet long, he can claim only 200 feet along the vein as his own. And if, moreover, he places his end-lines in such a position as to diminish still further his right to the possession of these 200 feet to their full depth, he certainly has no right to blame the law for his mistake. The law allows him full liberty to place his end-lines at any angle to the direction of his part of the outcrop.

B, on the same grounds, will own the part of the vein underlying the area, $m\ b''\ b\ q'\ m'$.

C will own the part of the vein underlying the area $c\ l'\ i\ c'\ g$.

D will own the part of the vein to the left of $m\ q$.

E will own all that part of the vein between the dip-lines $m'\ q'\ q''$ and $c\ l\ l'$.

A will also own that part of the vein underlying the triangular piece $c'\ i\ A$, because C, by locating his end-lines, as shown in the diagram, forfeited that part of the vein underlying the space between the intersecting lines $c'\ i\ l\ l'$ and $c'\ A\ x\ h$, and A owns by common law anything not excepted beneath his surface.

Those parts of the vein underlying the area $l'\ l\ i\ A\ x\ a\ g'\ h'$, and the area $q\ b''\ b\ q'\ q''$, belong to neither of the five locations, and are open to appropriation by surface-location to any one who first makes a valid location of these parts of the surface.

Any other interpretation of the law would lead to inextricable confusion, as shown in Mr. Raymond's able paper cited above, from which this hypothetical case has been taken. The law is plain and easy of construction and application, as soon as the identity of the true dip of a vein with the "downward course" of Section 2322 of the Revised Statutes is once recognized. But if there is anything in geological science about which there is no doubt or controversy, it is the definition of the dip or the downward course of any vein, bed, formation or stratum. The downward course of any vein is no questionable or optional direction, but can be mathematically ascertained by survey. The steepest pitch of the vein at any point of the vein is the direction of the dip of the vein at that point, and this direction is necessarily at right angles to the direction of the strike of the vein at the same point. Thus the line of dip from any point of the vein downward is easily ascertained by survey, especially as the main openings on any vein or lode generally follow the course of the vein on a level. In the few cases where any dispute as to

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the dividing boundary-lines of two adjoining extra-lateral rights to the same vein could occur, it would become thus simply a question of survey to establish the boundary-line of the extra-lateral right of the two adjoining claims on the same vein.

In most cases the boundary-lines between two adjoining extra-lateral rights to one vein would be two diverging lines, the one a dip-line, the other a prolonged end line, enclosing a wedge-shaped piece of the vein, to which really none of the compromising parties held any legal right of ownership, and which would belong to the party possessing that portion of the surface.

THE OCCURRENCE AND TREATMENT OF THE ARGENTIFEROUS MANGANESE-ORES OF TOMBSTONE DISTRICT, ARIZONA.

BY CHARLES W. GOODALE, BUTTE, MONT.

(NOTE.—Additional to the paper under this title, *Trans.*, xvii., 767.)

As an appendix to the above-mentioned paper, a drawing of the vertical projection of the Knoxville mine workings is here given.

In my paper it was stated that the ore-chimneys are found along a crack or plane of cleavage in the limestone; and as this crack is not parallel with the strata, it is safe to assume that the manganese was not deposited where it is now found before the tilting-up of the strata, but rather that the crack was caused by the folding, and that the formation of the ore was entirely subsequent to the change of position of the limestone.

There are here four chimneys, which have produced, as far as explored, over 12,000 tons of ore, containing at least 7000 tons of manganese oxides, and it is very difficult to apply Prof. W. P. Blake's theory of segregation from the surrounding limestone in accounting for the origin of this ore, as cross-cuts and drifts in the country-rock, even at short distances from the ore-chimneys, have not shown manganese in appreciable quantities.

It is easier to believe that, in the language of Mr. S. F. Emmons, in speaking of Leadville ore-deposits, there was a metasomatic interchange between the minerals brought in in solution and the limestone, and not a deposition in already existing cavities. This in-

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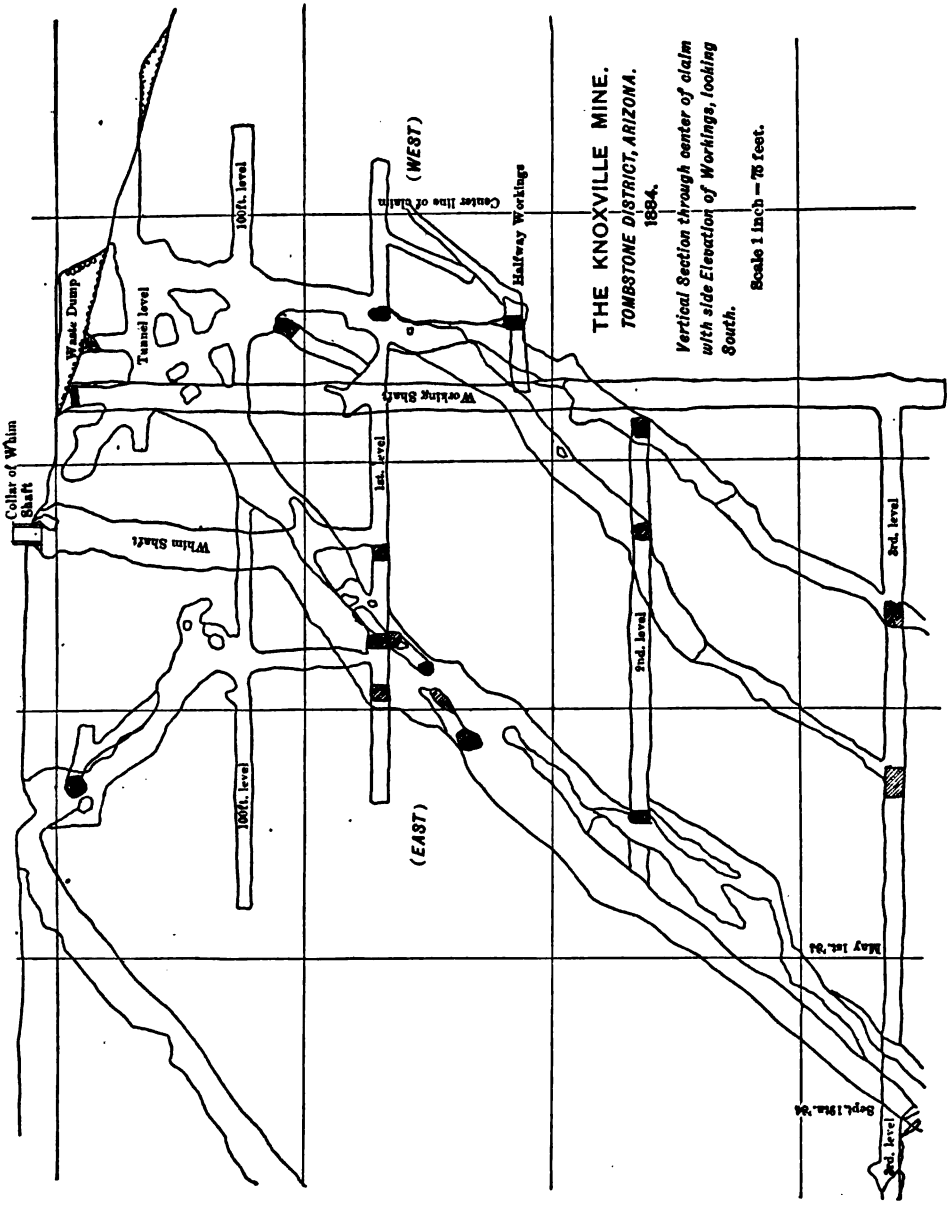
terchange or substitution must have taken place, according to the views of Mr. Richard Pearce, through the introduction of the man-



ganese as carbonate held in solution in carbonic acid waters, which dissolved the limestone and left rhodochrosite in its place.

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Since the discussion of my paper, search has been made in specimens of the Knoxville ore for rhodochrosite and for evidences of its crystallization, to confirm the theory of original deposition as carbonate; but no such indications have been discovered. Of course, the agencies which removed the carbonic acid may have destroyed all original forms.

After noting the effect of Knoxville raw and roasted ore on the quicksilver in the pans, we came to the conclusion that the earthy oxides of manganese were the troublesome minerals, and that hard, granular pyrolusite was quite harmless. The expulsion of water of composition from the wad and psilomelane in roasting, destroyed their earthy character, and with it their tendency to foul quicksilver.

In conclusion, I take this opportunity to correct one or two annoying errors, which remained in my former paper (*Trans.*, xvii.) by reason of the distance between author and printer, and the miscarriage of some communications by mail. Referring to the volume, the word "chloride," on p. 771, 3d line of 2d paragraph, should be *chlorine*. On p. 772, 8th line, the phrase "to keep the ore from falling," should be *to keep the ore falling;* and on the same page, 3d line from the bottom, "waste-building" should be "*roaster-building*. These errors occur on pages 5 and 6 of the pamphlet edition.

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[NOTE.—In this Index authors' names are printed in small capitals and the titles of papers in italics. Casual references, giving but little information, are usually indicated by bracketed page-numbers.]

ERRATA.

The following errors in the paper of Mr. Furlonge, beginning on page 334 of this volume, occurred by reason of the impossibility of waiting for the proofs to go to South Africa to be corrected by the author:

Page 336, line 7 from bottom, *microscopic* should be *macroscopic*.

Page 340, line 15 from top, *are* should be *are not*.

Page 340, line 11 from bottom, *microscopically* should be *macroscopically*.

Page 341, line 12 from top, *circular* should be *acicular*.

Page 347, line 13 from bottom, antimony *or* should be antimony *as*.

Footnote, page 529. *Composition of Various Commercial Aluminums.* Mr. A. E. Hunt desires to say that samples of metal recently received from the Aluminium-Industrie-Aktien Gesellschaft (works at Neuhausen, Switzerland, and Froges, Southern France), have been analyzed by Mr. Hendy, chemist of the Pittsburgh Testing Laboratory, with the following results:

	Fe	Si	Cu	Pb	K	Al
I.	0.03	0.53	none	none	none	99.44
II.	0.03	0.52	none	none	none	99.45

This company should, therefore, be added to the list of four given on pages 528 and 529; and a further addition is now required, namely, that of the Metal Reduction Syndicate, Limited, the works of which are at Patricroft, near Manchester, Eng.

Page 673. *Analysis of Aluminum-Bronze.* Dr. Simonds desires the statement to be added that the determinations of Al in this table are "by difference."

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